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ELEMENTS OF MINING

The quality of the materials used in the manufacture of this book is governed by continued postwar shortages.

Elements of Mining

GEORGE J. YOUNG
Member of the American Institute of Mining
and Metallurgical Engineers

FOURTH EDITION
Second Impression

McGRAW-HILL BOOK COMPANY, INC.
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ELEMENTS OF MINING

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THE MAPLE PRESS COMPANY, YORK, PA.

THIS EDITION IS DEDICATED
TO THE MINING FRATERNITY AND TO
THE MINING MACHINERY MANUFACTURERS.
THEIR COOPERATION MADE POSSIBLE
THE FOURTH EDITION.

PREFACE TO THE FOURTH EDITION

The objective of this book is to present coordinated starting points in mining practice that the student, mining man, and mining instructor may develop as their needs indicate.

The revision of the book was almost completed as the European war reached a conclusion. The war period placed unusual strains upon the resources of the mining industries and their personnel. The future of the mining industries will be in the direction of prospecting, exploration, and development of new discoveries to compensate for the drain upon reserves resulting from the war. New mineral deposits will be discovered. The demand for metals, fuels, and industrial minerals will increase as these components of modern life constantly grow in importance. This future challenges the young mining engineer to obtain a knowledge of the backlog of mining experience and to use it.

The perfection of mechanical equipment, some of which was introduced or tried in the interval of 1921–32, brought about improved mining practices. Notable in this development are the detachable drill bit, better rock drills, the drill carriage, drill-sharpening equipment, the light diamond drill and bort bit, heavy churn drills, side borers, rotary drills, undercutters, mechanical loaders, scrapers, conveyers, the axial-flow ventilating fan, autotrucks, tread tractors, shuttle cars, bulldozers, power shovels, the large scrapers, improved dredges and draglines, portable compressors, and diesel power units. Many useful accessories have also found application. The development of better explosives and a more widespread appreciation of their uses marked the 1932–45 period. The availability of highly developed mining tools and modern explosives as well as an understanding of their uses and limitations has increased labor efficiency and has been an important factor in cost control.

Acknowledgment is made of the use of articles in the *Engineering and Mining Journal* and other mining periodicals; likewise in the publications of the American Institute of Mining and Metallurgical Engineers, the Canadian Institute of Mining and Metallurgy, the Institution of Mining and Metallurgy, the Chemical, Mining, and Metallurgical Society of South Africa, and other technical organizations. To the authors of mining articles much credit is due not only for information but also for inspiring the author's attempt to weld this material into a condensed and informative book.

The cooperation of manufacturers of mining equipment in supplying catalogues and other information about their products is wholeheartedly acknowledged.

Particular acknowledgment is made of the assistance given by John P. Hutchins, Charles H. White, Correy Brayton, Herbert A. Sawin, Charles M. Romanowitz, George S. Dyer, J. H. Sackett, Raymond Mancha, Peter Ambrosiani, John B. Huttl, the late Professor Walter S. Weeks, and Charles A. Dobbel.

George J. Young.

Oakland, Calif.,
October, 1946.

PREFACE TO THE FIRST EDITION

The objective of "Elements of Mining" is to give the reader a comprehensive view of the mining problem. Fundamental engineering principles involving weight, mass, work, space, and the properties of the materials used and the rock masses encountered in mining have been given prominence. The dimensional data established by mining practice, cost analyses, and examples of cost are given in their proper places. The mechanical equipment in common use has been either illustrated or briefly described.

Coal mining has been included with metal mining, since while there are differences in the mining conditions the underlying principles are the same and one form of mining affords a contrast to the other. The relation of geology to mining has been brought out and accentuated wherever appropriate.

Exemplifications from mining practice have been reduced to a minimum and in most instances the information has been tabulated. In order to supplement this feature a limited bibliography has been selected and given under each subject.

The limitation plan on the size of the book has necessarily curtailed the number of illustrations and caused the condensation of the subjects of transportation and hoisting from two chapters, as originally planned, to a comparatively short statement. The fact that the publishers have in contemplation a separate book covering these subjects in connection with the larger subject of mine plant and equipment is a further justification for the brief treatment. The subject of mining law and examples of ore estimates were omitted because of the limitation of size.

In the preparation of the text I have drawn freely from the publications of the American Institute of Mining Engineers and the Lake Superior Mining Institute as well as other engineering societies and technical journals. In almost all cases acknowledgment has been made in the text. Much of the material dealing with coal mining has been taken from the bulletins of the Illinois Coal Mining Investigations. Illustrations of mechanical appliances have been taken from trade catalogues and the courtesy of the machinery companies in permitting their use is appreciated. To Dr. William H. Emmons I am especially indebted for reviewing the part of the text relating to geology. John E. Hodge supplied me with certain details of diamond drilling practice and Professor E. P. McCarty and J. J. Murphy were kind enough to furnish notes on the mining practice in Minnesota. Acknowledgment is made of the courtesy of many mine managers and superintendents who have facilitated my studies of mining practice.

George J. Young.

Berkeley, Calif.,
June, 1916.

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CHAPTER I

INTRODUCTION

The field of the miner and mining engineer concerns itself with the discovery and extraction of ores and naturally occurring mineral substances that are economically useful. For convenience the naturally occurring substances are divided into metalliferous ores, or ores of gold, silver, platinum, iron, copper, lead, zinc, antimony, manganese, tin, and tungsten; nonmetalliferous minerals, such as coal, peat, gypsum, infusorial earth, pumice, fuller's earth, limestone, clay, calcite, kaolin, feldspar, talc, soapstone, magnesite, dolomite, quartz, bauxite, garnet, corundum, phosphate rock, barytes, fluorspar, sulphur, pyrites, asbestos, graphite, asphalt, petroleum, gilsonite, salt, trona, niter, and borax; and building and ornamental stones, such as slate, marble, limestone, sandstone, granite, rhyolite, trap, travertine, opal, and gems.

DEFINITIONS

Ore.—The geologist and the mineralogist use this term in a restricted sense to apply to the minerals from which metals are obtained. The miner uses the term in the sense of any naturally occurring mineral substance from which a metal may be profitably extracted.

Mineral.—A general term, used to indicate any naturally occurring substance of definite composition and consistent physical properties. In a restricted sense the miner uses it to designate a valuable nonmetalliferous substance. Often the miner uses this term to indicate any naturally occurring compound with which he is not familiar.

Waste.—Rock or material removed in mining operations that contains either no metallic content or insufficient to admit of mining is called waste.

Ledge Matter.—A miner's term used to indicate the vein filling. It may or may not contain valuable minerals.

Gangue.—In but few cases do the metals occur native, and even when they do they are accompanied by a large amount of foreign material which must be separated from them and the handling of which entails expense. Gold, silver, platinum, and copper in native condition occur in important deposits, but all other metals and in many instances the foregoing metals occur in the form of chemical compounds. The ore minerals seldom are found alone, but have associated with them other minerals, usually valueless. These associated minerals, which are generally nonmetalliferous, are called gangue. The common gangue minerals are quartz, calcite, limonite, fluorspar, barite, siderite, rhodonite, garnet, feldspar, hornblende, and, in some instances, more or less altered country rock. An ore may be considered to be a mixture of one or more metalliferous minerals, in such proportion as to permit of profitable handling, and one or more gangue minerals.

Oxidized Ores.—Weathering and the action of surface waters alter most metalliferous minerals and convert them partly or wholly into oxides, carbonates, or sulphates. These compounds are characteristic of metalliferous deposits at the surface and oftentimes to a considerable depth below.

Sulphide Ores.—Ores below the influence of surface conditions and characterized by the presence of sulphides are termed either unoxidized ores or, if sulphides are conspicuously present, sulphide ores.

Mixed Ores.—Ores containing both oxidized and unoxidized minerals are called mixed ores.

Complex Ores.—This term has no precise meaning. It may signify an ore containing two or more metals; an ore difficult of treatment by reason of the presence of minerals which interfere with simple methods of treatment; or an ore composed almost wholly of several sulphide minerals.

Low-grade Ore.—A term used to indicate an ore that yields a small margin in profit.

High-grade Ore.—A term used to indicate an ore that yields a large margin of profit.

First-class Ore—Shipping Ore.—An ore of sufficient value to admit of selling to a smelter or reduction plant.

Second-class Ore—Mill Ore.—An ore that must be given some preliminary treatment such as concentration, picking, etc., before a marketable product or a product admitting of further treatment for the recovery of the metallic content can be obtained.

Ore Minerals and Nonmetalliferous Minerals.—The ore and nonmetalliferous minerals are described in treatises on mineralogy, and the student is referred to these.

UNIT VALUE AND TENOR OF ORES

Gross Unit Value.—The unit of weight taken is usually the short avoirdupois ton of 2,000 lb. For iron and manganese ores, bauxite, chromite, feldspar, and pyrites, the long ton of 2,240 lb. is in common use. The weight of metal per ton, as determined by assay or analysis, multiplied by the market price of the metal is the gross unit value of the ore.

Gross Recoverable Value.—In the treatment of ores by concentration, amalgamation, cyanidation, smelting, etc., only a part of the total metal present is recovered. The part recovered multiplied by the price is the gross recoverable value. The proportion recovered varies with the ore and with the method used. Examples are given in the following table.

TABLE 1.—APPROXIMATE PERCENTAGES OF METALS RECOVERED

Metal	Concentration	Amalgamation	Cyanidation	Smelting
Gold	80–90 (80–95)[a]	70–95	70–95	100
Silver	60–80 (80–95)[a]	60–80	60–90	95–98
Copper	60–80 (90–95)[a]	.....	.....	90–95
Lead	60–80 (80–95)[a]	.....	.....	90–95
Zinc	60–80 (80–95)[a]	.....	.....	80–90
Iron	60–75	.....	.....	95
Tin	60–80	.....	.....	90–95

[a] By flotation the higher percentages can be obtained in some cases.

Net Unit Value.—The difference between the gross recoverable value and the cost of mining, treating, and marketing the ore or products derived from the ore is the net unit value of the ore; in other words, the unit operating profit.

Constancy of Unit Values.—The ore in any given deposit varies in metallic content from point to point. In fact, it is very unusual to find uniform metallic content. This variability of content is one of the important factors that the engineer must constantly reckon with. In any given case the costs of mining, treatment, and marketing fluctuate between comparatively narrow limits, and their trend can often be predicted. Market prices of metals, excepting gold, also fluctuate. Percentage of recovery varies somewhat, but if there is considerable variation in the physical and chemical characteristics of an ore, this factor must be expected to show corresponding fluctuations. In systematic mining operations ores are carefully graded and sometimes the different grades are mixed by bedding, with the object of obtaining a uniform grade for the treatment plant. Average unit values are determined by systematic sampling and thus represent a known quantity.

Quantity. Size of Ore Deposits.—The quantity of ore in any mineral deposit determines its commercial importance. Unit ore values without the quantity factor have only a qualitative significance. The two factors, quantity and unit value, enable the gross value and the net value of a given deposit to be determined with a fair degree of accuracy. The gross value of a mineral deposit should be sufficient to cover the purchase price, the capital outlay for plant, equipment, development, operation, and marketing the product, as well as to leave an excess that measures the profit obtained from the enterprise. An enterprise must yield a profit to be commercially successful, and the larger the profit the more successful is the enterprise considered.

Both the quantity factor and the unit value are difficult to determine accurately, and their determination calls for care, judgment, and skill on the part of the engineer. A mineral deposit must be developed before these factors can be determined. By "development" is meant the sinking of shafts and the driving of levels and crosscuts in such a way as to delineate the outlines of the ore body at different altitudes. The average length, width, and thickness can be determined only in this manner. Systematic sampling enables average unit values to be determined. Some ore bodies may be delineated and samples of them may be obtained by systematic borings from the surface.

While it would be desirable in every instance to develop an ore body completely before extraction begins, it is not always feasible. In iron and copper deposits where surface boring is practicable almost complete information is obtainable, but with most gold and silver mines the two factors are known only for a part of the deposit. The principal element of uncertainty in mining enterprises arises from the difficulty of obtaining this information. The prevalence of gold and silver ores in shoots, of more or less irregular outline, distributed at irregular intervals along the strike and dip of the veins, renders it almost impossible for an engineer to know when he has exhausted the ore possibilities of a given mine. The frequent occurrence of parallel veins in close proximity to each other interjects another element of uncertainty.

The porphyry copper mines of the Western states and the iron mines of Minnesota present some of the best examples of successful estimation of the quantity and grade of ore in a given deposit or series of deposits in proximity. Table 2 gives estimated ore reserves and ore grades for a number of copper companies.

TABLE 2.—RESERVES OF COPPER COMPANIES
(*J. J. Croston, A.I.M.E.*)

Company	Reserves, tons	Per cent copper	Date
Kennecott—Utah	640,000,000	1.07	1930
Kennecott—Ely, Nev	77,000,000	1.22	1935
Inspiration Copper Co	69,010,770	1.37	1935
Miami Copper Co	84,624,860	0.94	1936
Phelps Dodge Copper Co	388,931,000	1.13	1932
International Nickel Co	205,590,592	2.00	1935
Chile Copper Co	938,931,000	2.15	1935
Andes Copper Co	112,549,287	1.47	1935

Other examples could be quoted, but these will suffice to exemplify some of the larger ore deposits and some of the most detailed engineering work in determining the limiting conditions of an ore deposit. These examples can be considered as representative of one end of the scale, at the other end of which the pocket mines of the West may be placed. The former require engineering skill, elaborate plants, and large investments of capital to place them on a commercial basis, while the latter require a skillful miner, a few drills, a forge, a wheelbarrow, and a small amount of dynamite. The amount of ore may vary from a few hundred pounds to several tons, while the value may run up to thousands of dollars. The large mines are characterized by continuous operation and the handling of enormous tonnages; the small pocket mines by intermittent operation and the handling of a small amount of material. Between these two extremes are found the many examples of mining ore deposits.

Tenor of Ores.—The economic conditions of any given mine determine the lowest grade ore which can be profitably handled. In general terms, the limiting grade of ore is that at which the gross recoverable unit value equals the total cost of placing the metal or metalliferous product from a unit of ore upon the market. At any one mine the tenor of the ore will vary during the life of the mine. Usually it is highest during the early years and lowest during the last years of a mine's existence. Within certain limits the policy adopted in mining will determine the tenor of ores. In bulk mining, the mine product may be low grade, the stopes being broken across their full width. In selective mining, the stopes may be restricted in both length and width and thus produce a much higher grade of ore. It is not always practicable to resort to selective mining, as the mineralization may be so distributed as to necessitate taking the whole ore body in mining operations. Where there is diffused mineralization, it becomes a matter of judgment as to where the stoping limits should be fixed. In most mines ore is simultaneously mined from a number of stopes, and usually the grade of ore will vary from stope to stope as well as from time to time in a single stope. By a system of stope samples this variation in grade can be determined and the drawing of ore from the different stopes can be so regulated as to send a product of approximately uniform grade to the mill or reduction plant. "Sweetening" the ore by drawing upon a rich stope in a mine is a practice often resorted to.

Price charts extending over a period of years exemplify metal price fluctuations. Supply and demand ordinarily are the chief determinants of prices, but general economic and political conditions play a part. The price at which a metal or mineral product can be sold depends upon wages and taxes plus a surplus to cover overhead, supplies, amortization of plant, depletion, capital return, and some provision for hard times. This is the seller's position; the buyer's position also is influenced by wages, taxes, and a multitude of conditions that establish the price at which he in turn can sell his products. Also, wartime price and control legislation has temporarily clogged the normal flow of basic products to buyers.

Mining properties fall into two broad classes, in one of which the grade of the ore is such that under most price conditions a good profit can be obtained; the other class comprises marginal properties, which can be operated only when prices are high.

One of the best examples of political influence is gold mining. Under the economic conditions prevailing up to 1929, gold mining gradually declined because the price of gold was fixed and wages, taxes, supplies, and other costs wiped out surplus in many instances. After 1929, wages and prices slumped as a result of depression, and the industry began to recover. From 1837 and up to 1932 inclusive, the selling price of gold was $20.67 per ounce, but in 1934 its price was finally established at $35 per ounce. World gold production was 19,585,000 oz. in 1929, 39,700,000 oz. in 1939. Domestic production was 2,208,386 oz. in 1929, 6,003,105 oz. in 1940, and 1,364,322 oz. in 1943 (estimated) after most of the gold mines had been shut down by a War Production Board order dated Oct. 15, 1942.

Another example of price influence is afforded by mercury mining. In 1929, 23,632 flasks of mercury were produced at an average price of $122 per flask; in 1943 some 53,500 flasks were produced, the average price being $195.21 per flask. The industry is often upon a marginal basis, but when the war demand shot prices up the industry responded by more than doubling production.

Naturally the student will want to know just what is the lowest grade ore that has been found practicable to mine. Many factors determine this minimum grade. The scale of operations, the efficiency and resourcefulness of the operating organization, the depth of mining, the presence or absence of large quantities of water, adverse conditions with respect to temperature and physical nature of rock masses and the distance from markets, smelting centers, and supply points, as well as other factors, will vary between wide limits.

Among the gold mines, Alaska Juneau maintained for many years a record position for mining low-grade ore. Over a period of 45 years, 70, 117,638 tons of mine run were produced, from which 32,586,720 tons were sorted as coarse tailing and 37,530,918 tons milled. Production for the period averaged 0.0518 oz. of gold per ton milled together with small quantities of galena concentrates; fine tailings averaged 0.0126 oz., coarse tailings 0.0089, and mine-run ore 0.0449 oz. per ton.

A total of 44 companies in the Transvaal mined 64,515,350 tons of gold ore in 1940. Total output was 13,535,504 oz., or 0.209 oz. per ton; maximum average yield of any one mine was 0.609 oz. and minimum was 0.12 oz. per ton. Working costs were 58.4 per cent and profit margin 41.6 per cent for the whole group.

The greatest gold mine, Crown Mines, Transvaal, South Africa, treated 3,558,000 tons in 1934 and produced 1,001,618 oz. of gold at a cost $24.52 per ounce, figuring the exchange rate at $4.90 per pound sterling and including charges and taxes. The Homestake mine of South Dakota produced 1,383,929 tons in 1936 with a recovery of 556,081 oz. of gold, 0.402 oz. per ton, at a cost of $21.69 per ounce. Carson Hill (California) produced in 1936 some 17,553 oz. of gold from 284,928 tons at a cost of $31.50 per ounce.

In 1941 Kennecott produced from all of its mines some 20.2 lb. of copper per ton as well as gold, silver, and molybdenite; of its four properties in the United States, the average assay of the ore was 1.04 per cent and production was 18.7 lb. of copper per ton. Phelps Dodge is now mining its Clay ore body by open pit. Estimates of its size, as computed from churn-drill holes, diamond-drill holes, and mine workings, indicated a tonnage of 230,000,000, averaging 1.06 per cent copper, with an estimated recovery of 18.2 lb. of copper per ton.

The lowest grade material handled in gold mining is exemplified in gold-dredging operations in California. The following table gives the dredging results obtained for a period of years by the Yuba Consolidated Gold Fields operating at Hammonton, Calif.:

TABLE 3

Year	Yardage, cu. yd.	Average yield, cts. per cu. yd.	Total yield	Operating costs, cts. per cu. yd.	Surplus over cost
1915–16	17,750,000	12.02	$2,133,578	4.27	$1,374,868
1919–20	23,523,000	13.60	3,213,213	6.99	1,645,548
1928–29	24,948,295	8.81	2,196,846	4.53	1,066,124
1935–36	35,969,934	9.79[a]	3,522,126	4.49	1,615,139
1936–37	37,135,623	11.71[a]	4,351,919	4.72	1,754,587
1938–39	48,881,544	9.26[a]	4,528,426	4.37	2,137,922

[a] Gold at $35 per ounce; others at $20.67.

The student should review the reports of a number of mining companies and note the percentage content of the ore, operating costs, tailings loss, and recoveries. In this way he may visualize the individual conditions and sense the limiting margin that distinguishes profitable from unprofitable enterprises.

VALUE OF METALS, ORES, AND NONMETALLIFEROUS MINERALS

The economic disturbances of the First World War unsettled metal prices greatly; late in 1920 and 1921 metal prices were extremely low and resulted in drastic curtailment of mining operations in the United States and elsewhere. The economic disturbances of 1929–32 also caused a great recession in metal prices along with greatly lessened mining activity. Average annual prices, given in Table 4, reflect the fluctuations over a sufficient period to indicate the uncertainties of mining with respect to the influence of metal prices. In the early stages of the Second World War efforts were made to stabilize prices of metals, but as neither taxes nor wages were likewise controlled the position of mining in 1942 continued uncertain. Under the wartime demands, however, production of ores, minerals, and metals reached a record height. The prices of silver, copper, lead, zinc, quicksilver, gold, and other metals were controlled.

SALE OF ORES AND MINERAL PRODUCTS

Price.—The product of a mine may be metal, ores, concentrates, minerals, and certain by-products. These are marketed to the United States Mint, smelting companies, refiners, mineral dealers, and manufacturing companies, each of which may be interested only in a particular product. For example, the Mint is interested only in gold and silver; custom smelters are interested in gold, silver, lead, copper, and zinc ores and concentrates of these metals; certain mining companies produce refined metals and sell directly to manufacturers; industrial mineral producers sell to the trade interested in such minerals. The journey from producer to consumer often involves several transfers, each of which raises the price. The *Engineering and Mining Journal* publishes basic metal and mineral prices each month. Many ores and minerals are sold by producers under contracts. Ore- and mineral-buying practice varies in different localities and with different ores and industrial minerals.

Gold Ore.—The sale of gold ores to milling, smelting, and ore-buying companies follows long-established practices. The market for gold is at present closely controlled by governments, the price in the United States being fixed at $35 per ounce. Custom-milling companies pay for an agreed-upon percentage of the gold and silver content of an ore, the percentage depending upon the gold extraction obtainable from the particular ore as determined by laboratory testing or by experience. A milling charge to cover operating costs, marketing, overhead, and profit is deducted and the balance is paid to the seller. At Cripple Creek, under the $20.67 gold price, gold ore was paid for at $20 per ounce of gold content, less a treatment charge which was a minimum of $4 per ton for ore 0.5 oz. or less and a maximum of $6 per ton for ore 1 to 1.5 oz. per ton. Freight charges were deducted. In the palmy days of Goldfield, Nev., 90 per cent of the gold and silver content of a milling ore was paid for at $20 per ounce for gold and the prevailing rate for silver less $7 per ton for milling charges; freight was also deducted.

TABLE 4.—AVERAGE PRICES OF PRINCIPAL METALS, 1906–44[1]

Year	Copper, electrolytic standard		Lead		Tin		Zinc			Silver		Antimony, New York	Quicksilver, New York	Aluminum, New York	Platinum, New York
	New York	London	New York	London	New York[a]	London[b]	New York	St. Louis	London	New York	London				
1906	19.278	87.282	5.657	17.370	39.819	180.646	6.198	6.048	27.020	66.791	......	21.730	40.90	35.75	28.04
1907	20.004	87.007	5.325	19.034	38.166	172.638	5.962	5.812	23.771	65.327	......	14.840	41.50	45.00	30.98
1908	13.208	59.902	4.200	13.439	29.465	133.124	4.726	4.578	20.163	52.864	......	8.004	44.84	28.70	16.32
1909	12.982	58.732	4.273	13.042	29.725	134.774	5.503	5.352	22.185	51.502	......	7.466	46.30	22.00	24.87
1910	12.738	57.054	4.446	12.920	34.123	155.308	5.520	5.370	23.050	53.486	......	7.386	47.06	22.25	32.70
1911	12.376	55.973	4.420	13.970	42.281	192.353	5.758	5.608	25.281	53.304	......	7.540	46.54	20.07	43.12
1912	16.341	72.942	4.471	17.929	46.096	209.420	6.943	6.799	26.421	60.835	......	7.760	42.46	22.01	45.55
1913	15.269	68.335	4.370	18.743	44.252	201.679	5.648	5.504	22.746	59.791	......	7.520	39.54	23.64	44.88
1914	13.602	61.524[c]	3.862	19.076[c]	34.301	156.564[c]	5.213	5.061	22.544[c]	54.811	......	8.763	48.31	18.63	45.14
1915	17.275	72.532	4.673	22.917	38.590	163.960	13.230	13.054	67.533	49.684	......	30.280	87.01	33.98	47.13
1916	27.202	116.059	6.858	31.359	43.480	182.096	12.804	12.634	72.071	65.661	......	25.370	125.49	60.71	83.40
1917	27.180	124.892	8.787	30.500	61.802	237.563	8.901	8.730	52.413	81.417	......	20.690	106.30	51.59	102.82
1918	24.628[d]	115.530	7.413	30.100	[e]	330.138	8.159	7.890	54.180	96.772	......	12.581	123.47	33.53	105.95
1919	18.691	90.796	5.759	28.590	63.328	257.601	7.338	6.988	42.879	111.122	......	8.190	92.15	[e]	114.61
1920	17.456	97.480	7.957	37.832	48.273	295.866	[e]	7.671	44.372	100.900	......	8.485	81.12	32.72	110.90
1921	12.502	69.356	4.545	22.752	28.576	165.265	[e]	4.655	25.845	62.654	......	4.957	45.462	.....	75.033
1922	13.382	62.123	5.734	24.097	32.554	159.450	......	5.716	30.003	67.521	34.406	5.471	58.946	.....	97.618
1923	14.421	65.840	7.267	27.147	42.664	202.148	......	6.607	33.058	64.873	31.926	7.897	66.502	.....	116.537
1924	13.024	63.149	8.097	34.421	50.176	248.737	......	6.344	33.728	66.781	33.969	10.836	69.761	.....	118.817
1925	14.042	61.920	9.020	36.429	57.893	260.974	......	7.622	36.624	69.065	32.088	17.494	83.128	.....	119.093
1926	13.795	57.971	8.417	31.075	65.285	291.016	......	7.337	34.105	62.107	28.686	15.988	91.903	.....	113.269
1927	12.920	55.653	6.755	24.192	64.353	288.953	......	6.242	28.513	56.370	26.046	12.393	118.159	.....	84.636
1928	14.570	63.703	6.305	21.060	50.427	227.131	......	6.027	25.284	58.176	26.746	10.305	123.506	.....	78.580
1929	18.107	75.416	6.833	23.246	45.155	203.850	......	6.512	24.790	52.993	24.460	8.956	122.145	.....	67.655
1930	12.982	54.611	5.517	18.077	31.694	141.873	......	4.556	16.570	38.154	17.666	7.667	115.000	.....	45.358
1931	8.116	38.342	4.243	12.958	24.467	118.375	......	3.640	12.215	28.700	14.594	6.720	87.351	.....	35.665
1932	5.555	35.962	3.180	11.913	22.017	135.848	......	2.876	13.545	27.892	17.843	5.592	57.592	23.30	36.455
1933	7.025	36.359	3.869	11.670	39.110	194.510	......	4.029	15.666	34.727	18.144	6.528	59.227	23.30	30.993
1934	8.428	36.319	3.860	10.935	52.191	230.273	......	4.158	13.657	49.973	21.229	8.901	73.865	23.30	36.465
1935	8.649	35.430	4.065	14.238	50.420	225.473	......	4.328	14.082	64.273	28.952	13.616	71.992	20.00	34.150
1936	9.474	42.650	4.710	17.599	46.441	204.445	......	4.901	14.920	45.087	20.075	12.240	79.917	20.00	42.926
1937	13.167	59.339	6.009	23.326	54.337	242.133	......	6.519	22.258	44.883	20.067	15.355	90.180	19.92	51.773
1938	10.000	45.411	4.739	15.266	42.301	189.457	......	4.610	13.990	43.225	19.523	12.349	75.470	20.00	35.901
1939	10.965	....	5.053	......	50.323	226.177	......	5.110	......	39.082	20.570	12.359	103.940	20.00	36.748
1940	11.296	....	5.179	......	49.827	256.425	......	6.335	......	34.773	22.281	14.000	176.870	18.70	37.924
1941	11.797	....	5.793	......	52.018	......	......	7.474	......	34.783	22.493	14.000	185.023	16.50	36.000
1944 (May)	11.775	....	6.500	......	52.000	....	......	8.250	......	44.750	23.500	15.839	115.538	15.00	35.000

[a] 99 per cent tin.

[b] Prices of tin, London, from 1906 to 1914 from statistical report of Metallgesellschaft, Frankfurt am Main.

[c] Averages of 9 months, no quotations being made during August, September, and October.

[d] Average of 11 months, no quotations being made in December.

[e] No average computed.

Quotations for copper, tin, zinc, antimony, and aluminum, in New York or St. Louis, are in cents per pound. Up to 1929 quicksilver price was based on a 75-lb. flask; subsequently a 76-lb. 1-oz. and then a 76-lb flask became standard. Silver and platinum are quoted in cents and dollars per ounce respectively. All London quotations are given in pounds sterling per long ton

[1] *Engineering and Mining Journal.*

Most gold mines mill their own ores and produce gold bullion, concentrates, cyanide precipitates, and minor by-products. Gold bullion can be sold to the Mint, or all the products to the custom smelters and refiners. In California, gold-bearing concentrates are purchased and delivered to the smelting plant on the basis of gold content, as follows: From 0.03 to 0.5 oz. gold at $19 per ounce plus 90 per cent of the excess of current price ($34.91) over former price ($20.67); 5 oz. and less than 10 oz., $19.50 per ounce plus 90 per cent of excess current over former price; 10 oz. and less than 15 oz., $19.75 plus 90 per cent excess of current price over former price; 15 oz. or over, $20 plus 90 per cent of excess of current over former price. A minimum deduction of 0.5 to 1 oz. is made from the silver content and the balance is paid for at the government fixed price of 71.11 cents per ounce (July 6, 1939) for domestically produced silver. Smelter charges are from $6.45 to $10 per ton. On very small lots sampling charges are often deducted. Cyanide precipitates and refinery slags are purchased under other terms.

Smelter Schedules.—Copper, zinc, lead ores, and concentrates are purchased under various schedules that determine the details of settlements, charges, and deductions. Each custom smelter has its own schedule, but many of these schedules conform to similar patterns. Usually a settlement date is made, such as date of assay return, and the price of the metals is of that date according to *Engineering and Mining Journal* quotations. The smelter purchases ore in many lots of various sizes, and a period of varying length intervenes between the purchase and the final marketing of the metals purchased. To protect themselves from fluctuating metal prices, most smelters endeavor to sell each day an amount of metal equivalent to their purchases for that day in the form of ore. Some ores, by reason of accessory metals, are difficult to treat and may entail greater than average metallurgical losses or costs. The smelter reimburses itself by exacting penalties and extra charges. For ores which are very easily treated or which may be necessary for more favorable smelting mixtures, extra credits may be given or premiums paid. Some schedules also endeavor to provide for wage increases. Thus many schedules are complicated. A few examples are summarized.

Silver Ores.—Such ores are purchased subject to a minimum deduction of 0.5 to 1 oz. per ton, 95 per cent of the remaining silver being paid for at the domestic or foreign price of silver, depending upon origin; treatment charges are from $5 to $6 per ton for lots of 10 tons or more and $10 per ton for lots less than 10 tons; the grade of ore is also a factor, and treatment charges go to $10 per ton for high-grade silver ores.

Copper Ores and Concentrates.—Copper concentrates are settled upon the basis of wet assay, minus 10 to 20 lb. per ton at the New York price for electrolytic copper minus 2.5 cents per pound; treatment charges range from $3.50 to $7.50 per ton; freight is deducted and a sampling and handling charge is made for small lots of ore. Silver content if above one ounce is paid for at 95 per cent of assay at Mint price if the ore is of domestic origin, and at Handy and Harman quotation if of foreign origin. If gold content exceeds 0.03 oz. per ton, all the gold is paid for at 92 per cent of Treasury price, or $32.20 per ounce.

Lead Ores and Concentrates.—Lead concentrates are settled upon the basis of wet lead assay minus 1.5 units, times 90 per cent; payment at New York prices minus 1.5 cents per pound; basis 30 per cent lead dry assay. For less than 30 per cent deduct 10 cents per unit for each unit under 30 per cent (a unit is 1 per cent); for each unit over 30 per cent add 10 cents. Treatment charges range from $2.50 to $6 per ton. Silver, if over 0.5 oz., is paid for at 95 per cent of content at New York quotation; gold, if over 0.02 oz. per ton, is paid for at $31.818 per ounce up to 5 oz. and $32.318 per ounce if over 5 and up to 10 oz. per ton. Various penalties are in force: bismuth, arsenic, and antimony are considered objectionable and are penalized, arsenic and antimony being allowed one unit free and all above carrying a penalty of 50 cents per unit; zinc from 6 to 8 per cent is allowed free and all above is penalized at 30 cents per unit; sulphur is allowed free up to 2 per cent and all above is charged 25 cents per unit to a maximum charge per ton of $2.50; iron is given a credit of 6 cents per unit and insoluble a penalty of 10 cents per unit.

Zinc Ores and Concentrates.—Payment for zinc ores is made on the basis of 82 per cent of the zinc content at St. Louis price less 29 cents per 100 lb.; 75 per cent of gold content in excess of 0.05 oz. per ton is paid for at $32.32 per ounce; 75 per cent of silver in excess of 2 oz. per ton is paid for at Mint price; 65 per cent of lead in excess of 2.5 units is paid for at market price less 1.5 cents per pound; 75 per cent of copper in excess of 2 per cent is paid for at electroylitic price less 5 cents per pound; base treatment charge is $28 per ton, plus or minus $2 for each one cent up or down in the per pound price of zinc. Freight is deducted and provision for wage increase is made in smelter settlements.

Table 5.—Antimony Prices per Short Ton Unit

Per cent antimony	Price per unit	Per cent antimony	Price per unit	Per cent antimony	Price per unit
40	$1.40	47	$1.48	54	$1.515
41	1.405	48	1.50	55	1.520
42	1.410	49	1.50	56	1.525
43	1.415	50	1.502	57	1.530
44	1.430	51	1.503	58	1.535
45	1.450	52	1.505	59	1.540
46	1.465	53	1.510	60	1.550

Antimony Ores.—Antimony ores in 1941 were purchased by a few companies in carload lots f. o. b. shipping points, at rates shown in Table 5. Ores of more than 35 per cent and less than 40 per cent antimony content were also purchased, but at lower rates.

Miscellaneous Ores and Minerals.—Iron ore is sold on the basis of its iron content. The presence of silica in excess of a low percentage diminishes the price, and in Bessemer iron ores the phosphorus must be less than 0.10 per cent. Hard iron ores as compared with soft pulverulent ores command a better price. Manganese ores are sold on their manganese content. The silica limit is 8 per cent, the phosphorus limit 0.25 per cent. Above these percentages penalties are exacted. Nonmetallic minerals are sold on the basis of chemical analyses and certain impurities are limited in amount. Where this amount is exceeded the mineral must be cleaned or penalties are exacted. In some cases where the degree of purity necessary cannot be obtained the mineral is unmarketable. Grades and prices are given in the market reports of the *Engineering and Mining Journal* for a number of different ores and minerals.

MINING INDUSTRIES

Mining Industries of the United States.—The data of the mining industries in the United States, according to the 1939 census, are given in the table on page 13.

MINERAL DEPOSITS

A classification of ore and mineral deposits is given in the outline shown on page 14.

GENERAL FEATURES OF DEPOSITS

Outcrop.—The outcrop is the edge or surface of a mineral deposit that appears upon the surface. It may be definitely and plainly visible or almost obscured by surface detritus.

Wall.—The upper wall of an inclined vein or deposit is called the hanging wall; the lower, the foot wall. Walls may be definite or indefinite.

Dip and Strike.—The dip is the maximum angle of inclination downward that a vein or bed makes with a horizontal plane. The strike is the course of the vein measured at right angles to the dip and referred to the true or magnetic meridian. Both dip and strike are used in describing veins and beds. They are not constant features but may vary from place to place in any one vein or bed. In irregular deposits these terms have no equivalent, but the course of the deposit is the course of the major axis, and the dip is the dip of the major axis in a vertical plane at right angles to the horizontal major axis.

Overburden.—The detrital or other material overlying a flat or moderately inclined deposit is called the overburden. In many cases it

Mineral Industries of the United States, 1939

Industries	Number of operations	Number of mines, quarries, and wells	Salaried officers and employees	Wage earners (average for year)	Value of products
All operations	20,927	361,202	82,809	779,032	$3,430,258,644
Coal:					
Anthracite	346	507	5,411	82,822	189,647,913
Bituminous	5,009	5,686	19,656	369,156	727,357,537
Crude petroleum, gas	7,782	347,645	30,322	105,166	1,375,953,576
Metals:					
Copper	35	51	2,908	23,844	141,634,842
Iron ore	100	177	2,228	20,137	150,872,108
Lead	62	76	998	6,984	31,467,413
Zinc	138	170	974	8,653	31,184,092
Gold, lode	820	841	1,612	17,279	86,063,020
Silver	150	163	368	4,244	19,715,727
Gold, placer	306	339	477	3,228	28,026,824
Mercury	64	61	74	602	1,830,116
Molybdenum	5	5	112	910	15,410,581
Tungsten	35	49	134	960	3,353,852
Stone:					
Limestone, T	965	1,256	2,129	25,619	80,655,601
Granite, T	199	242	384	4,417	12,876,081
Basalt, T	101	120	278	1,910	9,658,219
Slate, T	70	79	115	1,341	4,162,547
Marble, T	31	44	41	1,405	2,708,857
Sand and gravel	1,253	1,563	2,818	16,959	79,402,277
Miscellaneous:					
Abrasives, natural	33	41	45	366	1,295,228
Asbestos	9	9	9	160	492,487
Asphalt, natural	23	23	123	730	2,968,145
Barite	37	47	62	792	2,065,048
Bentonite	27	29	62	357	1,982,129
Clay and shale, T	833	1,061	760	10,648	24,847,153
Feldspar	47	59	54	512	981,162
Fluorspar	60	61	109	1,287	3,397,624
Fuller's earth	21	22	116	562	2,106,721
Gypsum	34	59	97	1,327	4,568,925
Magnesite and brucite	3	4	12	216	1,396,168
Mica	22	21	20	190	326,573
Phosphate rock	33	40	382	3,372	12,286,471
Potash	5	5	284	1,516	13,963,561
Pyrites	6	5	15	189	601,588
Rock salt	17	17	181	1,380	6,896,271
Sulphur	8	10	507	1,517	31,812,230
Talc and soapstone	29	38	167	970	3,269,087
Tripoli	9	12	20	139	426,761
Vermiculite	7	7	8	56	149,883
Producing operations	18,920	361,040	77,019	736,150	3,221,927,057
Fuels, total	13,345	353,969	57,509	566,956	2,392,753,928
Metallic ores, total	1,753	1,992	10,110	88,394	515,008,693
Stone, total	1,521	1,929	3,158	37,287	117,033,887
Miscellaneous, total	439	526	2,330	15,906	92,881,119

Note.—T signifies the total, including closely related producing operations under the designations. Miscellaneous total does not include clay and shale but does include several minerals not presented in the table.

The foregoing table was derived from Table I, vol. I, Mineral Industries, Sixteenth Census. In the same volume see Tables X–XXX, and other tables, under particular mining industries.

is thin enough to warrant its removal and to mine the deposit by open pit or quarry. Coal, iron, placer deposits, gold, copper, and other metals are frequently obtained in this manner.

Physical Nature of Walls and Ore or Mineral.—Walls and ore may be consistently the same throughout a deposit, but this is unusual and considerable variation in the physical characteristics may be expected

Classification of Ore and Mineral Deposits

		Metalliferous	*Nonmetalliferous*
I. Deposits due to weathering and erosion	*In situ* or close proximity to source	Iron ores Manganese ores Gold Tin Platinum	Residual clays Bauxite Phosphates Barite Ochers
	Eroded and transported to a distance	Gold placers Tin placers Platinum placers Monazite and rare earths Magnetite	Detrital clays Quartz sand Fuller's earth Gem stones
II. Deposits concentrated in bodies of surface waters	By precipitation	Bog iron and manganese ores Oölitic iron ores Carbonate iron ores	Limestone Chalk Dolomite Phosphate beds
	By evaporation		Salt Gypsum Sodium sulphate Sodium bicarbonate Borax Potash minerals
	By sedimentation		Peat Lignite Coal Diatomaceous earths
III. Deposits concentrated by circulating ground waters	Derived from rock masses	Iron ores Ores of copper, lead, vanadium, zinc	Barite Sulphur (?) Magnesite Talc Soapstone Asbestos Gypsum Sodium nitrate Nitrates Borax
	Derived from primary deposits	Ordinary and sulphide enrichment Copper, gold, silver	
IV. Deposits formed by ascending heated waters associated with igneous intrusions and gases		Ores of mercury, antimony, gold, silver, lead, tin, zinc, copper, tungsten, molybdenum	Quartz Calcite Alunite Borax (?)
V. Deposits due to metamorphism, closely related to IV		Ores of copper, iron, lead, zinc, tin	Garnet Graphite Corundum
VI. Deposits formed in magmas, closely related to IV	Segregations	Magnetite Chalcopyrite Arsenopyrite Platinum Cassiterite Chromite Nickeliferous sulphides	Corundum Diamond
	Pegmatite dikes	Cassiterite Wolframite Columbite Molybdenite Rare minerals	Feldspar Mica Apatite Lepidolite Zircon Gem stones

and should be made a matter of study and observation. Methods of excavation are modified to suit the nature of the material handled.

Methods of support can be more intelligently devised if the physical peculiarities of the walls are known. Too often an engineer neglects detailed study of this factor and embarrassing difficulties arise as a consequence.

Interruptions to Continuity.—Deposits may pinch out locally on both the strike and dip or may be divided by a fault. Some deposits are faulted to a marked extent, and considerable difficulty is caused not only in following the deposit but also in working it. The presence of many faults indicates general weakness in a rock mass and calls for certain precautions in working it.

Folds.—Bedded deposits, particularly sedimentary deposits like coal, may be folded to such an extent as to offer serious difficulties in applying customary methods. The expense of working such a deposit may reasonably be expected to exceed that required for a regular deposit.

Vertical Range and Depth below the Surface.—As an element of importance in working a deposit, its vertical range from highest to lowest point and depth should be known. The selection of hoisting apparatus and the development and working may be considerably influenced by this factor. The vertical range in the Calumet and Hecla mine approximates 5,000 ft. from the surface. The copper mines of Butte, Mont., are worked at depths ranging from 1,000 to 4,000 ft. The Kennedy mine in California is opened to a vertical depth of 5,000 ft. The disturbance of the equilibrium of large rock masses by mining through a deep vertical and a wide lateral range is a factor that must be reckoned with and provided for in the early stages of a mine.

Ore Shoots.—No ore deposit shows uniform distribution of metallic content. Approximate uniformity of distribution is shown in the auriferous reefs of the Transvaal, the amygdaloidal copper lodes of the Lake Superior district, the porphyry copper mines of Nevada, Utah, Arizona, and New Mexico, the Homestake mine, and the iron mines of Minnesota and Michigan; but the great majority of gold, silver, gold-and-silver, copper, silver-lead, and lead-zinc deposits show an erratic distribution of metallic content. Where the values are concentrated sufficiently for extraction, that part of the deposit is called an ore shoot. The shape of the ore shoot is variable, and the size ranges from small bunches or pockets of ore up to many thousands of tons. The primary cause of ore shoots is the existence of favorable conditions for deposition in the zone occupied by the shoot. A shoot caused by enrichment is deposited at or close to the ground-water level and continues along the course of the deposit. Ore-bearing solutions may be obstructed by an impervious stratum and, rising along this, may deposit in proximity to the stratum. The chemical character of the wall rock may be such as to cause precipitation in certain areas. The intersection of fissures is often a favorable location for deposition, particularly where two solutions differing chemically come together. Each locality, and in fact each deposit, presents its peculiar problems, and only extended observation can bring out the facts to assist the miner. The known erratic character of the occurrence of ore shoots is sufficient to warrant the miner's belief that every vein or even slightly mineralized zone is the possible receptacle of an ore shoot until proved otherwise.

Pay Streak.—In alluvial deposits the gold is usually concentrated near the bedrock. This portion of the placer is called the pay streak. In some few cases, at a higher level within the gravel deposit, another pay streak may be found, but this is unusual. The width and trend of the pay streak are variable. Gold is most likely to be deposited where the current of a stream decreases. A bar often forms on the concave side of a bend, and gold may be found on the bedrock and within the material deposited in the bar. The head of a filled basin or the head of a delta, where the stream debouches upon a coastal plain, is also a favorable place. These observations apply to recent placers. Old placer deposits are often such that former topographical conditions cannot be determined with any certainty, and consequently there is little to do but dig to bedrock.

Water.—Water is not of uncommon occurrence in mineral deposits. The handling of it always entails expense. Its source, amount, and chemical nature are matters for the investigation of the engineer. The possibility of sealing off ground water or of deflecting surface sources of supply may well repay the time spent upon its consideration. In humid or semihumid regions, ground water may be expected to occur within the first 50 ft. and to extend often from 500 to 1,000 ft., as a vein or mineral deposit is often a favorable channel for the accumulation and movement of ground waters. In arid climates ground water may not be encountered in considerable quantities within 300 to 500 ft. of the surface. At Tonopah the first deep shaft sunk was practically dry at 900 ft. from the surface. Many very deep mines are dry in their lower levels. In mineral regions where extensive postmineral fracturing due to faulting has taken place, accumulations of water in the fractured zones introduce a difficult problem. The Comstock mines from a depth of several hundred to 3,200 ft. presented a water problem of this nature. It is true that in this case many flows of hot water were encountered in the lower levels, but most of them were handled by moderate-sized pumps, and the accumulations of fissure water gave the most trouble.

Underground Temperature.—Although the increase of ground temperatures with depth is generally of no serious moment, in some deep mines high temperatures have been encountered that brought about difficulties in the working of the deeper levels. The temperature gradient under different conditions is shown in the accompanying table.

Gradient in Feet for Each Degree Centigrade Rise in Temperature[a]

Unaltered rocks	114.1
Underneath high ridges and mountains	150.0
In or near eruptive rocks	46.0–79
In vicinity of hot waters or where chemical processes of decomposition are active	33.0–56
In coal mines and borings in coal-bearing strata	76.0–115
In mines after ventilating currents have cooled off rocks	102.0–377

[a] Condensed from Lindgren's "Mineral Deposits."

Underground Gases.—Special precautions and methods are required in the mining of deposits that liberate combustible or irrespirable gases. Many coal beds are characterized by the presence of marsh gas and related gaseous hydrocarbons. Nitrogen and carbon dioxide are occasionally encountered in metal-mining operations, but their presence is usually rare. Marsh gas has also been found, but again this is unusual.

Surface Topography.—The study of a deposit with reference to the surface topography has for its object the selection of the point of attack. The layout of a quarry or open pit is necessarily dominated by the position of the deposit, but nevertheless access and drainage must be provided. An economical approach can be found only by a careful study of topographical conditions. The selection of a site for a surface plant, working shaft, or adit allows somewhat more freedom, and wisdom shown in its careful selection may eliminate dangers from snowslides, cloudbursts, and other contingencies.

General.—The normal stages in the history of a mining enterprise are discovery; exploration to determine features of the deposit and the grade of the mineral; development for the purpose of accurately determining its geometry, average and variations in mineral grades, and preparation for working; equipment for mining, handling the mineral, and preparing it for market; the orderly working of the deposit on the basis of a daily tonnage; the extension of development in advance of mining requirements; and the continuation of exploration having for its purpose the proving of extensions or the finding of new deposits in contiguous areas. Working continues, except when interrupted by adverse market conditions or other difficulties beyond the power of the management to control, until the commercial mineral has been exhausted, at which point operations cease and the plant and equipment are dismantled and as much as possible is sold.

The working life of a mineral property obviously depends upon the quantity of commercial mineral found and the rapidity of its extraction. A maximum rate of working is desirable, but this is often restricted by the available working capital, by the size of the plant, by the market demands, and by the inherent difficulties of operation. Thus 100 tons per day may supply the demand within a radius where freight rates exclude a competitor. Other producing areas more favorably situated with respect to freight rates and consuming centers may supply the remaining market. On the other hand, demand may already be adequately supplied and the new producer may have to fight for his market, so that a long time may be necessary before he can develop a sufficient market for his product. If he has some advantage in freight rates, cost of production, or quality of product, he can compete successfully. Skillful sales management may compensate in part for any lack of commercial advantage. Metals occupy a less competitive position than most nonmetallic minerals, in that custom smelters are usually ready to buy and to treat ores and concentrates. Large refining companies purchase crude petroleum from producers. Coal-mine owners compete for their markets. The gold miner is the only producer free from competitive strife since the price of his gold is fixed.

Available capital is a large factor in determining the magnitude of initial operations. Considerable skill is required in the planning of development, equipment, and operation so as to keep within conservative limits and at the same time to have sufficient capital in reserve to provide for the interval between initial production and realization from sales of the product. When the money is exhausted, additional capital must be secured with which to complete plans and to provide sufficient operating money until the enterprise can be made self-supporting upon the basis of the sale of its product. Many small and some large enterprises encounter difficulties at the start by too ambitious plans and too small a purse. Good engineering involves careful planning, accurate estimates of costs, and the securing of results within the capital limits. Unforeseen conditions and accidents may interfere with the best of plans, but the farseeing engineer will always make some provision for contingencies of this kind.

How capital is raised for a mining enterprise may be summarized as follows: First is the time and effort of the prospector as well as the money he expends in his activities. As he is seldom in a financial position to advance to the exploration and development stages, almost invariably he invites friends and others to advance the necessary money, giving them an interest on a partnership basis, or failing in this he attempts to sell his prospect upon the initial showing. In the event of a sale into strong hands, the capital requirements may be advanced with which to explore and try out the possibilities of the prospect. In other instances, the preliminary partnership may be reorganized upon the basis of a company and a portion of the company shares may be sold to others, the capital raised being expended upon exploration and development as well as for equipment and facilities. Comparatively few such enterprises succeed in reaching the operating stage (producing regularly) unless the capital thus raised is wisely expended and the work is so efficiently done as to serve its destined purpose.

Not all prospects turn into operating mines, so that stockholders in such enterprises are in the position of venturing their capital. The blanks outnumber the successes. The large operating company in its search for mines spends money freely but wisely and counts upon finding a commercial property among the number it may explore. Nevertheless a relatively large sum may be expended under competent direction and with all the geological skill possible without winning a prize. Probably a large proportion of exploration companies fail, but this may be due to the fact that the talent for finding and recognizing mines is not widespread. An exploration staff, otherwise competent, may lack this essential talent, which is admittedly hard to find. However, mines have been discovered by the application of geological principles. They will continue to be discovered more and more by geological studies in conjunction with geophysical methods.

Commercial Deposits.—Numerous mineral deposits can be found and many are known, but a deposit to be commercial or exploitable must meet certain requirements, among which are quantity and average value that will yield a margin over operating expenses, a workable ore or mineral, and comparative freedom from such restrictive measures as might be enforced as a result of the pollution of streams or the air. Water is essential, and the cost of obtaining a suitable supply might be prohibitive in some cases. The development of power and its cost needs careful investigation. The source and supply of labor and the limitations imposed by weather, by altitude, and by topographic environment are also given consideration. Access by roads and transportation conditions are studied.

The absence of a known method of ore treatment does not necessarily constitute an insurmountable obstacle but is rather a challenge to metallurgical skill and resourcefulness. New methods of ore treatment and new methods of cleaning and handling mineral have been successfully worked out as is shown by Trail, Ajo, and Chuquicamata, and in the case of the Chilean nitrates and Searles Lake potash. Research, experimentation, and inventive power may be necessary, but the trail of the miner is marked by achievements of metallurgists and chemists who have come to his assistance. Given a sufficient quantity of ore or mineral of commercial value to provide the incentive, it is seldom that failure is recorded. New problems will continue to arise, but the background of past achievement and experience is always available.

Property.—Control of mining property may be by deed, by lease and bond, or by agreement based upon some royalty. The first comprises outright ownership of the land and its appurtenances, together with such water rights as may be attached to the property. The second comprises a conditional ownership for a certain period, which may be converted into complete ownership by the fulfillment of the conditions of the bond and lease, usually by completing certain money payments. The third is based upon the operation of the property for a given period and the payment of a royalty or a percentage of the gross value of the mineral taken from the property. An additional control is that exercised by a mineral-claim owner, who enjoys possession as long as he complies with the assessment requirements of the federal mining laws and such provisions as may be found in the state law. Leases may be given upon part of a mining property, and these are usually conditional on the payment of a percentage of the gross proceeds resulting from the sale of the ore or mineral sold by the lessee. Actual ownership of a mining property can be determined by a search of the county property records. Liens recognized by law are also determined from county records. Claims not of public record should be shown in the records of the company or individual owning the property. Payment or nonpayment of taxes against a property may be found in county records. These are matters of importance where a mineral property is to be purchased. The validity of a deed should be determined by a competent attorney.

Neighboring Properties.—Boundary mining properties frequently involve difficulties with respect to the use of the ground surface for roads and waste-rock dumps and, through the operation of the apex right, to the ownership of ore bodies. It is for this reason that neighboring properties are purchased and consolidations of mining properties are brought about. A farseeing management will secure by purchase near-by property which may be necessary for operating facilities or which might serve as a source of future embarrassment.

Governmental Contacts.—Water rights, rights of way, utilization of water power, taxation, and the abatement of public nuisances, such as smelter smoke, pollution of streams by tailings or in other ways, bring the mine owner in contact with federal, state, and county governments. Where specific statutes regulate such matters, the individual or the company becomes responsible for their observance. Mining camps may be subject to housing and sanitary regulations. Mine operation may also be subject to specific regulations that have for their purpose the safeguarding of the worker or providing compensation in the event of his injury. State inspection of mines is to be expected, and the reporting of accidents to the mine inspector is usually provided for by specific regulations. In addition, mine management is called upon to supply information relative to production, numbers of employees, and other subjects to government statistical divisions. Working of mining claims within the limits of national forests brings the operators under the jurisdiction of the Forestry Service and its regulations. The taking up of mineral claims and mill sites under the federal laws brings the claimants into contact with the Land Office when claims are to be patented. County, state, and federal government have therefore to be considered in relation to the operation of a mining property, and the management must therefore be informed as to the limitations originating from statutes or from regulations imposed upon the industry by governmental departments.

Engineering.—An engineer observes conditions, gathers data, makes measurements and surveys, and then visualizes what is wanted. His experience and knowledge enable him to make or to supervise the preparation of plans and details as well as to select the machinery to accomplish certain objectives, which may be a mine surface plant, a quarry layout, a metallurgical plant, a working shaft, or the development of a mine. The more completely plans are prepared, the more systematically can construction be accomplished and costs controlled. There is an orderly sequence that may be followed in construction, and this, when skillfully planned, results in the efficient use of labor, materials, and time. Materials must be purchased, transported, and properly stored in advance of requirements, so that when they are wanted no delays will result. The construction superintendent, foremen, and workers of different kinds must be recruited and brought to the job. Living quarters and commissary are prepared in advance of their coming.

Sequence charts for materials, for handling labor, and for construction can be prepared and are based upon the plans, details, and specifications of the work. The endeavor is made to coordinate the work so that the objectives may be attained at a specified time. Each part of the problem is analyzed by itself and then in relation to the whole. Parallel operations, distinguished from sequential operations, are carried out simultaneously, so that the end results of each parallel operation fit into the major sequence at the proper time and without delay. A construction job thus involves a major sequence and parallel minor sequences coordinated with the major sequence and anticipated by an orderly stream of materials and equipment both for construction and as provided in the plans.

Time and its efficient utilization reflect the ability of the engineer. Delays result from vague and incomplete plans, inability to visualize and to analyze, unusual weather, poor supervision, inaccurate estimation of time schedules, failure to obtain deliveries of materials, and sometimes from hard luck or the incidence of unforeseen conditions. The selection and supervision of construction crews are vital factors in obtaining efficiency and reducing delays to a minimum.

Some mining engineers specialize upon design and construction of mining plants, some upon the operation of mining properties, and others act in consulting capacity to companies or to individuals interested in mining. Banks and other financial institutions often employ mining engineers to pass upon mining securities, to examine mining properties, and to determine the financial status of mining companies. Mining engineers occasionally enter the field of mine promotion and the financing of mining properties. The field of the mining engineer is a broad one, and there are opportunities for specialization in the major divisions of the industry as in coal, iron, nonferrous metals, nonmetallics, placer mining, petroleum, and in construction materials as sand, gravel, and stone. In these major divisions there are opportunities for design, construction, equipment, for operation, and for consultation work.

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CHAPTER II

PROSPECTING

Definition.—The fundamental meaning of the term "prospecting" is search, and the objective is discovery. It is sometimes loosely used to cover the operations incident to the determination of the boundaries of a mineral deposit. The surface discovery of a mineral deposit, and more particularly the metalliferous deposits, does not give an adequate conception of its extent, and as a consequence more or less underground work is necessary. The term "exploration" may be applied to this extension of the discovery work. When exploratory work shows the deposit to be of importance, the subsequent work of defining the limits or boundaries and preparing it for the extraction of the mineral is termed "development."

Surface and Underground Prospecting.—Surface and underground work are the two natural divisions of prospecting. The two methods of surface prospecting may be termed direct and indirect. In the direct method the prospector endeavors to find an outcrop or the evidences of the existence of an outcrop in what he considers a favorable locality. Outcrops may be pronounced, obscure, or hidden. The "float," or detached ledge matter resulting from the erosion of a deposit, may be found close to its origin or at a considerable distance away. The size of the pieces of float, the angularity or smoothness of the pieces, and the proportion of the pieces with respect to other material are of importance in approximating the distance from the outcrop and the size of the outcrop.

It is evident that the application of this method must involve a great deal of work of a negative character. The limited area of surface exposures, the obscurity of most outcrops, and the relatively small amount of the debris (float) resulting from the erosion of these exposures are the factors that prevent the rapid discovery of all the outcropping mineral deposits. Thus accident may play a larger part than deliberate search. The announcement of a discovery, particularly of one that appears important, has the effect of stimulating search in the immediate vicinity with the result that often other discoveries are made.

The "favorable locality" of the prospector is in or near a mining district that is either producing or in the exploratory stage, or else in a locality that "looks as if it might contain mineral." While the expression is indefinite, the frequently marked characteristics of the area in the vicinity of a known ore deposit justify the intelligent prospector in the use of the term. In other words, the surface soil and often the rock exposures about most ore deposits look different from those at a distance. In searching for an outcrop the prospector should systematically examine the whole area and particularly the unusual and obscure rock exposures. The beds of ravines and gullies are favorable places, as often rock exposures are clear of debris and soil. The crest of ridges affords another area from which alluvial and soil material is frequently absent. Float, if present at all, is most likely to be in stream beds and ravines. If it is found, the general method is to trace it up a ravine until no more can be discovered. The sides of the ravine are then searched and the float is traced up on one side or the other until no more can be found. If no visible outcrop occurs, trenching is resorted to. In hunting for float the material removed from the holes of small animals should be carefully examined. Diligent use of a pick and the breaking of soil-covered lumps of rock often reveal pieces of float.

The indirect method presupposes a knowledge of structural geology and of the occurrence of mineral deposits. The more important types of rocks and rock alteration should be familiar to the prospector. The valuable minerals likely to be encountered should be carefully studied with the object of readily recognizing them. The method is briefly that of elimination. If we suppose a given area to be under examination, elimination can be effected by considering topography, rock associations, geological age, structural peculiarities, and type of deposit. The area under consideration may be completely eliminated, or the greater part may be eliminated as unimportant and the remainder classed as the critical area. The purpose is to restrict the detailed search to critical areas rather than to examine a large area indiscriminately.

Topography.—A land area may admit of subdivision into two important parts, one in which erosion is active and from which a portion or all the surface products of weathering are removed, the other upon which the products of erosion are deposited. Naturally the former includes the topography of moderate to high relief, while the latter includes the topography of low relief. If the rate of erosion is greater than the rate of surface weathering, exposures of rock in place will result, while if the reverse is true, soil and detritus will accumulate and mask the rock exposures. As a consequence regions of high relief show many surface exposures of rock in place, while regions of intermediate relief are covered with soil and debris of varying thickness. The grade of a drainage system will vary from zero in the valleys to steep slopes in the mountains. The transporting power of a stream is dependent upon the velocity of the current, and velocity is dependent upon the grade of the stream bed. Boulders and stones are therefore deposited in the upper reaches of the stream, coarse gravel in the intermediate, fine gravel in the lower, and sands and very fine gravel in the flood plains of the stream. Volume of flow also plays a part.

Mineral deposits are either alluvial, talus, or "in place." In prospecting for recent placers, search would be confined only to those areas where deposition is or was active, thus eliminating all topography of intermediate to high relief. As placers are secondary deposits they could originate only in the stream systems that dissect formations bearing the valuable minerals; hence areas of alluviation containing placers are limited to certain favorable localities. It should be noted that in an extensive stream system varying conditions of grade are met with, and we may expect to find local deposition of alluvial material at places where the grade flattens. Where the source of the primary minerals is intersected and is confined to a restricted area, it is needless to say that placer deposits may not be expected above this source.

Prospecting for placers is somewhat complicated by the fact that they were formed not only in the present epoch but also in the Quaternary and Tertiary periods. Modern erosion has left Quaternary gravel deposits as benches, often well above the level of the present stream terraces, while segments of old Tertiary river beds have been exposed by erosion at considerably greater elevations. In some cases the conditions are reversed and the Quaternary and Tertiary deposits might be found beneath the present alluvial deposits. Many of the Tertiary gravel deposits of California were covered with lava flows and are now contained in the ridges and hills between which the present streams flow. Erosion in certain instances has exposed the edges and ends of these deposits and distributed the gravel down the hillsides. The prospector discovering this gravel can readily trace it to its source. Sometimes the old stream bed can be seen in an erosion scarp. Flat terraces, well above modern stream terraces, and old beach lines are worthy of the attention of the prospector in regions where modern placers have been discovered and worked. Beach placers also are sometimes found.

Deposits in place can be most readily found where the relief is moderate to high. Beck states:

> The oldest German writers made a distinction among mountains. They preferred gentle middle mountains and flat valleys to abrupt, jagged, piecemeal mountains of Alpine character, and on the whole not without reason. Modern folded mountains in a geological sense, such as the Swiss Jura, the larger part of the Alps, and the Carpathians proper with their abrupt forms, contain, on the whole, but very scanty ore deposits; on the contrary, the old mountains, especially those whose principal uplifts fall within the Paleozoic period, such as the Erzgebirge and Harz, are on the whole much richer in ore deposits, and the same is true of the old mountains planed down by the activity of water.[1]

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[1] R. Beck, "The Nature of Ore Deposits," translated by W. H. Weed, p. 662.

While in a general way this observation is valid as applied to some of our own mining districts, the topographic criterion must not be taken by itself; geological conditions must also be considered. Prospecting for metalliferous deposits is seldom attempted where alluvial or talus deposition has deeply covered the rocks. A noteworthy exception may be made to this statement. In Minnesota many of the deposits of iron ore are covered with a deep layer of glacial drift. The topography is subdued, varying from flat to gently rolling. The few surface exposures gave the key to the situation and, in spite of the heavy surface covering, the discovery and delineation of many other deposits were made by test pits and boring. Sedimentary deposits are not infrequently covered by surface soil and no near surface exposures are to be found. The presence of such deposits can be discovered only by boring or deduced only by geological reasoning from distant exposures and a study of structural conditions. The key to the stratigraphical problem, where surface exposures are absent, must be sought either in the neighboring hills where erosion and uplift have exposed the formations or in the deep dissections caused by erosion.

Rock Associations.—In a general way certain associations of rocks and mineral deposits have been noted. Sweeping generalizations concerning such associations must be viewed with caution. If mineral-producing localities are considered individually, valuable generalizations often can be made and are made. Specifically, coal is associated with sandstones, limestones, shales, and clays; petroleum with limestones, shales, soft sandstones, and slightly consolidated sands; magnesite with serpentines; garnet with metamorphic rocks; and phosphates with limestones and shales. Among the metals, gold ores are associated with quartz, quartzite, granite, slate, schist, diorite, andesite, rhyolite, andesitic tuff, rhyolitic tuff, limestone, etc.; copper ores with quartz, granite, monzonite, diorite, and limestone; lead ores with quartz, calcite, limestone, and quartzite; zinc ores with calcite, limestone, and quartzite; iron ores with cherts, quartzites, and slates; tin ores with granites and the minerals tourmaline and lepidolite; nickel and nickeliferous copper ores with gabbros, diorites, peridotites, and serpentines.

Siliceous gold and silver deposits are sometimes divided into two groups, *Tertiary* and *pre-Tertiary*. The distinction is especially important. The Tertiary is associated with igneous rocks of Tertiary age and is in close association with local centers of igneous action belonging to that period. The deep-seated types are associated with igneous rocks of pre-Tertiary age, and where associated with sedimentaries, such sedimentaries are Cretaceous or older. Sedimentary rocks not in proximity to igneous rocks of pre-Tertiary age are unfavorable for the presence of precious metal deposits. Where they are in juxtaposition to igneous rocks their presence must be considered only as an incidental association. Undisturbed or only slightly disturbed sedimentaries are evidence of the absence of intrusive rocks and freedom from structural movements. Where deposits are genetically connected with igneous rocks, we must expect them to be present in areas characterized by these rocks. Where they owe their origin to the action of underground waters not connected with igneous intrusion, we must expect the deposits to occur in proximity to the formations in which the minerals are disseminated and from which they were derived. Where they are of metamorphic origin we must expect them in metamorphic rocks, and where they are of sedimentary origin we must expect to find them in areas occupied by sedimentary formations.

The principal rock groups are surface igneous, intrusive igneous, deep-seated igneous, metamorphic, and sedimentary. A given area where rocks of one or more of these groups occur may be divided accordingly, and deposits which are characteristic of one or the other should be sought for in the appropriate area.

Another important consideration is rock alteration. It has been observed that where solutions of deep-seated origin have been at work they have in many cases attacked the enclosing wall rocks of fissures and have caused more or less extensive alteration. Sericitization, chloritization, kaolinization, and silicification are effects produced by hot fluids rising from deep-seated sources. Surface rock areas that show these types of alteration are important. Kaolinization and decompositions due to weathering simulate and sometimes mask the effects of primary alteration. One noteworthy distinction can be made: Primary alteration is in many cases restricted to small local areas which often follow a given general direction and which are comparatively narrow, whereas weathering is generally widespread. The almost universal presence of pyrite in close association with ore minerals, the readiness with which this mineral weathers, and the highly colored nature of its residue often mark the weathered area in the immediate vicinity of an outcropping deposit and serve to distinguish it from the areas that show the alteration due solely to weathering. Pyrite occurs in small quantity in most igneous rocks, and when weathered may usually be distinguished from possible ore-bearing ground by the absence of other alteration products of primary mineralization.

Special studies of leached outcrops, first made by Charles H. White and Augustus Locke, and later by Roland Blanchard and P. F. Boswell, have resulted in the development of a technique for observation and a method of interpretation of the observations that are of value in indicating not only the presence of ore bodies at depth but also in teaching something of their nature. Roland Blanchard states that most leached outcrops can today be translated, with more or less assurance, into terms of the sulphides or other minerals from which they were derived. However, it is necessary to map areas carefully on the basis of painstaking sampling and thorough examination of the material by hand lens, microscope, and sometimes by chemical analysis. Slight differences in structure, color, mineralization, and density afford the principal basis upon which discrimination is made, and when these are mapped certain areas can be distinguished and the critical areas can be selected. Locke in his book, "Leached Outcrops," gives details of his methods. Roland Blanchard and P. F. Boswell, in an article in the *Engineering and Mining Journal*, Feb. 18 and Mar. 3, 1928, give numerous examples of the significance of outcrops and their variations.

Geological Age.—The prospector's expression "favorable formation" is a recognition, crudely expressed, that certain rocks and series of rocks are more favorable than others. The accumulated facts of geology afford much that is of assistance to the prospector. In the search for sedimentary deposits geological age is of importance, but it is less so where metalliferous deposits are concerned. The work of geologists has brought out a number of important generalizations concerning the occurrence of mineral deposits in certain geological periods.

It is evident that in all contemporaneous (syngenetic) deposits geological age is significant, while in all subsequent (epigenetic) deposits the age of the rocks becomes very much less significant. If the period of mineralization can be definitely determined, all younger formations in a given area can be eliminated. It should be further noted that each locality and region has its peculiar associations, and where these are known or can be determined by study they become the criteria for the elimination of unimportant areas. Geological time limits are determined by the examination of plant, shell, and animal remains and, where these are absent, by the stratigraphical relation to strata of known age.

Structure.—Uplift, faulting, sheeting, and folding are of significance to the prospector. Syngenetic sedimentary deposits like coal may be looked for in undisturbed or only slightly disturbed rocks, but epigene ores of deep-seated origin can rise only through fissures, or breaks in the crust. As deep dissection of mountain masses by erosion facilitates detection of mineralized areas, elevated regions where the crust has been broken may be favorable places for finding ore deposits.

Faults are of three classes: Those which have a genetic connection with ore deposition and which have provided the channel for the circulating solutions; premineral faults, which have no bearing upon the problem; and postmineral faults, which might serve a useful purpose in exposing an ore deposit or might lessen the value of a mineral deposit by dividing it into widely separated parts. A much-faulted area is not necessarily favorable, while a comparatively undisturbed area is apt to be unfavorable for epigenetic ore bodies. Faults which are not mineralized and which do not intersect a vein are not significant.

Zones of sheeting may be sometimes of considerable importance. Frequently the direction of a dominant system of sheeting planes is also the direction of faulting. In some cases deposition has been influenced by the intersection of sheeting planes and faults. In an area under examination, sheeted rocks or shear zones should be scrutinized.

The occurrence of gas and petroleum along the axes of anticlines causes this structure to be of special importance to the oil prospector. Faulting also is of importance, as this structural feature is often imposed upon the former.

Types of Deposits.—A classification of the different types of ore deposits has been given in the preceding chapter. The characteristics of each type must be learned from experience and study. The search for a particular kind of a deposit in any given area is simplified by knowledge of its characteristics.

Miscellaneous.—The presence of a spring within a critical area is sometimes indicative of proximity to a fissure and thus may be the means of discovering a deposit. The significance of a spring can be determined by detailed study of its relationship to structural features. The deposition of mineral compounds by the spring waters should attract special attention. In most cases the relationship of a spring to a deposit is obscure, but nevertheless the prospector should look upon it as a possible indication of proximity to a deposit and seek to discover its importance. Tertiary river deposits may sometimes be discovered by the fact that they are often underground watercourses and that dense vegetation results where the water reaches the surface and forms a spring. In many cases the vegetation may mask the spring, but its very presence in an unusual place is sufficient to attract the attention of a keen-eyed prospector.

Vegetation usually masks outcrops and is an interference rather than an aid to the prospector, though in cases where there is a marked differentiation in the distribution of plant life, close observation in a given region may elucidate facts of value. The tracing out of a given formation by observing the presence or absence of tree growth is one of the possibilities. Beck mentions particular plant growths as characterizing certain outcrops, as for instance the calamine violet upon calamine outcrops; the *Amorpha canescens* upon limestone soils above galena beds; in Montana the *Eriogonum ovalifolium* about silver veins. While the subject of the relationship of plant growth to mineral-bearing formations is of no little importance, it must be said that no thorough study has been made of it, and until this is done its value for prospecting purposes is an open question.

Each of the two methods of surface prospecting, the direct and the indirect, is by itself useful. Both methods should be applied in any given case. It is evident that the application of the elimination method requires the careful examination of the area. Critical areas may thus be discovered, and the problem then reverses itself into an intensive examination of such areas. The discovery of new mineral deposits will without much doubt be made by the careful examination of known critical areas or by the application of a method, similar in principle to that described, for the discovery of new critical areas. The successful prospector of the future will be one trained to make effective use of the accumulated knowledge of the geology of mineral deposits.

Underground Prospecting.—In a narrow sense this term includes the driving of underground excavations from the workings of a known deposit. Extensions of a known ore deposit may be looked for upon the direction of its strike or dip. Ore bodies may lie within a certain formation, which thus becomes the favorable zone and in which shafts, drifts, and crosscuts may be driven. Veins are frequently paralleled by other veins and systematic crosscutting at different depths may disclose them. Diamond and core drilling may be utilized in place of the ordinary workings. The careful study of the underground and surface geology and in particular of the habit of known ore shoots is a necessary preliminary to systematic prospecting of this nature.

In a broader sense, all excavations made for the purpose of discovering ore or mineral are included in the term. Necessarily such workings are as far as practicable confined to critical areas. The promiscuous sinking of shafts and driving of crosscuts and drifts can be looked upon only as a waste.

The simplest working is the "surface trench." This is in the nature of a crosscut and should be driven transversely to the trend of the formation that may carry ore or serve as a guide to ore. It must expose the underlying formations. The complete removal of surface debris from a critical area is attempted only under special conditions. In the Cobalt District (Canada) the surface soil in one instance was entirely removed by hydraulicking from an area under examination, and in another instance a large lake was drained and the silt and mud were removed. The narrowness and richness of the veins in that district were sufficient justification. The placing of surface cuts requires some judgment. The underlying principle should be to make each cut represent as great a length as possible. This requires the placing of limits upon the length of the area. The first cut is made in the middle. If no discovery is made, the next two cuts should be placed so as to bisect the halves, and so on. Sometimes the thickness of the cover is the chief consideration in selecting the position of the trench. When the cover is too thick, recourse must be had to a crosscut tunnel or a shaft and crosscut. The placing of the shaft should follow the principle described for trenching. Where crosscuts intersect a vein or promising leads, drifting along these follows. Once a lead has been struck it is advisable to have the exploratory workings, whether shafts or drifts, follow this lead.

Where the cover is thick, the area extensive, and no special indications worthy of following are found, boring is the most convenient method to apply and is quicker and in most cases less costly than shaft sinking and crosscutting. In placing boreholes the same general principle described before holds. In the absence of any special surface indications that might give preference to one part as compared to another, the whole critical area is delineated and a borehole is sunk in the center. This operation is continued until the nature of the underlying formations has been determined or negative evidence has been obtained. The center transverse line is divided on either side of the central bore. Boreholes are sunk at these points. If the evidence is negative, the longitudinal axis is divided in the same manner and two additional holes are sunk. Squares may be laid out from the parallel lines intersecting the boreholes and additional holes sunk at the corners. In prospecting work upon the Mesabi Range, which exemplifies the conditions named before, five boreholes are sunk at distances of 500 ft. in a 40-acre tract, and if no ore is obtained the next 40-acre tract is treated in a similar manner. Where ore is discovered the tract is divided into 100-ft. squares, and additional holes are sunk at the intersections of the 100-, 200-, or 300-ft. ordinates. In prospecting a gravel deposit for the pay streak, transverse sections are made, the first section being a mid-point upon the length. Holes on this section are spaced 50 to 100 ft. apart. The use of boreholes for prospecting placer, iron, copper, lead, zinc, sedimentary, and saline deposits has received extensive recognition and is a convenient and successful method. Diamond-drill and core boring are made use of for prospecting in relatively hard formations. More or less success has attended their use for finding deposits of the precious metals. Extensions of known ore bodies or new ore bodies in proximity to known ore bodies are frequently sought for by these methods. The use of the diamond drill for deep bores of 4,000 to 10,000 ft. for proving the extension of known gold formations in the Transvaal, South Africa, is a good example.

Examination of Rocks, Alluvial Material, and Minerals.—In surface examinations, alluvial, talus, and gulch dumps claim the prospector's attention. While float is often found in such material, the bulk of it will afford information concerning the nature of the formations from which it was derived. It is thus possible quickly to find out many of the formations in a mountain range or upon a given watershed without conducting a laborious trip over it. Material from rock outcrops should be broken and fresh rock surfaces compared with weathered. Tests for hardness with a knife, acid tests, and the use of a magnifying glass enable the principal rocks to be readily determined. Where the material is finely divided a portion of it should be panned for heavy minerals and these, if present, should be determined. Concentration with miner's pan, horn, or batea is one of the most convenient tests. Unusual rock outcrops and float should be tested by crushing a portion and panning. The following table shows some of the minerals that may be found in this manner.

Table 6

Color of Concentrate	Minerals Apt to be Present
White or gray	Cerussite, angelsite, barite, scheelite
Sparkling white	Zircon
Yellow	Tungstite, molybdite, wulfenite, monazite
Yellow-metallic	Gold
Red	Cinnabar, proustite, pyrargyrite
Purplish	Cerargyrite
Dark	Hematite, hubnerite, wulfenite
Dark red	Garnet
Dark brown	Cassiterite, hematite, hubnerite
Black, shining	Magnetite, specularite
Black, dull	Argentite
Dark, metallic	Platinum and metals of the platinum group

The fluorescent properties of some minerals, notably scheelite, are an aid to their identification in the field. Scheelite is often difficult to identify, as it resembles a number of common minerals. Its high specific gravity helps in identification, but where it is disseminated in relatively low percentages it is generally overlooked. The ultraviolet light excites the fluorescence of scheelite, which glows with a bright blue light when pure, with a white light where accessory molybdenum is less than 0.5 per cent, and with an increasingly yellow light as the percentage of molybdenum increases. The presence of scheelite even in small percentages is readily determined by the use of the Mineralight, a portable lamp of special design for field use. This useful prospector's aid enables a speedy and accurate examination of outcrops, veins, and underground workings. Many valuable scheelite deposits have been discovered by its use. Quantitative estimates can be made by determining the proportion of areas of fluorescence to total surface areas examined. This ratio multiplied by 1.2 gives the percentage of tungstic acid (A. W. Jolliffe and R. E. Folinsbee). Such minerals as calcite, halite, calomine, sphalerite, and others also show fluorescence of various colors, or sometimes phosphorescence.

GEOPHYSICS

Until the advent of geophysics the discovery of mineral deposits was restricted principally to such deposits as were exposed upon the surface or whose presence could be deduced from geological studies of structure and by the identification of favorable strata. The geologist rather than the prospector took the leading part. Particularly in the search for petroleum the geological deductions became more and more important. The cumulative knowledge that resulted from the number of petroleum geologists engaged in this work rapidly built up a geological practice that is a valuable asset to the industry. Nevertheless, geological methods could not meet all the conditions, nor did the geologists in the metal-mining field find their methods comprehensive enough to meet the needs brought about by covered deposits that showed no clues as to their presence. Something more specific than speculation based upon general geological studies was needed. The diamond drill and the churn drill can be used to test speculative inferences, but the probabilities of failure are large and the cost limits it seriously.

Probably the first instrument used in testing physical properties of the earth's crust in place was the dipping needle, by which variations in the earth's vertical magnetic component could be determined. Variations in compass directions as a result of magnetic masses, or local attraction, had long been noted. The dipping needle proved successful in facilitating the discovery of magnetic-iron ore bodies. Likewise, the measurement of variations in the horizontal component of the earth's magnetic field indicated the position of abnormal areas. Better and more precise instruments were developed, such as various types of magnetometers and the earth inductor. In recent years a remarkable development of other methods has taken place. Electrical conductivity has been investigated in various ways. Variations in the gravity component, the measurement of the velocity of sound waves in various crustal media, and the measurement of radioactivity have been investigated, and both instruments and technique have been developed for the measurement of these properties of the earth's crust. A departure from normal, in other words an anomalous condition, is the basis of all these methods. The deduction of the cause of the anomaly, whether it be an ore body, a structural change, a petroleum deposit or other mineral mass, or an unusual condition not resulting from such occurrences, is the objective. Upon the accuracy of this deduction hinges the success of geophysical prospecting. Usually the determination of an anomaly with reference both to position and to depth is insufficient; it must be interpreted with reference to all the geological facts that can be obtained about structure. In addition, boring is necessary to confirm or disprove the theory that has been deduced.

A considerable degree of success has been obtained with geophysical methods, and some of their limitations are now known. A background of geological as well as instrumental experience seems to be necessary for the successful interpretation of the measurements. This may be the combined effort of instrument man and the geologist, but the latter must be a thorough student of the geophysical methods that may be used. The cost of examining a given area by geophysical methods varies widely, depending on the method used and the thoroughness of the work. Although the unit cost based upon the acre may not appear great, the aggregate cost of most geophysical surveys is high enough to limit their application.

Magnetic Methods.—The magnetic susceptibility of minerals is widespread, but only a small number of minerals possess it in amount sufficient to be of use in their detection. The ferromagnetic group—native iron, magnetite, pyrrhotite, franklinite, and ilmenite—are the principal minerals in either massive or disseminated form that react sufficiently to influence the magnetic needle when they are close by and in sufficient quantities. Most rocks have little susceptibility, but rocks that have a moderate content of magnetite or ilmenite, or both, will react, though not so strongly as the pure mineral. Olivine gabbro, gabbro, serpentine, and to a lesser degree basalt are in this last-named group. The magnetic susceptibility of specularite is pronounced; that of chromite and hornblende is less evident. Although various tables of magnetic susceptibility have been published, there seems to be considerable discrepancy in the values ascribed to the more important magnetic minerals, and there is a need for a scale of the magnetic properties of rock minerals. Metalliferous minerals, apart from the iron group, are so feebly susceptible as not to admit of detection, unless they are associated with minerals of considerable magnetic susceptibility. At first sight it would appear that the scope of magnetic methods would be restricted to the detection of magnetic ore bodies or hematite, with which is associated magnetite, but there is a wider field. For example, placer deposits characterized by concentrations of black sand (magnetite) can be detected. A still wider field is the study of formations associated with the more basic rocks or containing significant amounts of magnetite and ilmenite.

Only three instruments will be described. The first instrument used to detect ore bodies by their magnetic reaction was the miner's dipping needle. This consists of a magnetized needle swinging upon pivots in a vertical plane and with a graduated circle not quite in the plane of rotation. A balancing weight is placed upon the south portion of the needle so as to balance the downward pull of the magnetic north. The needle is balanced so as to be horizontal or in a position normal to the magnetic dip. The adjustment of the instrument is made in a position where the earth's magnetic field is normal or free from local disturbances. Only relative changes in the earth's magnetic field are indicated. Readings of this instrument are taken after it has been oriented in the magnetic meridian and the needle has been released. The needle is read upon the swing. Some observers place the dip needle in a plane at right angles to the magnetic meridian (see Henry Louis, *Mining Magazine*, July, 1921, p. 19) upon the theory that in this plane the needle will be affected only by the magnetic ore body or other local anomaly. The chief virtues of the dip needle are its convenience and the rapidity with which observations can be made. It is not sufficiently sensitive to detect small anomalies, and it is impracticable for quantitative determinations of the vertical magnetic intensity. An exhaustive study of its limitations has been made by Noel H. Stearn in "Geophysical Prospecting," 1929, p. 345.

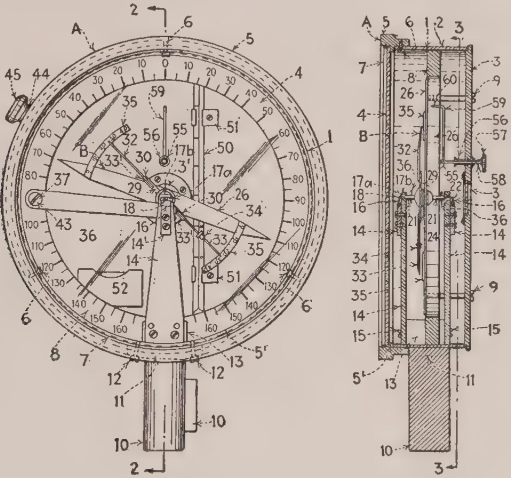

FIG. 1.—Hotchkiss Superdip magnetometer.

The Hotchkiss Superdip magnetometer was devised to overcome the limitations of the dipping needle. Its sensitiveness can be adjusted. The principal parts consist of a bar magnet of special tungsten-cobalt steel so mounted as to swing readily in a vertical plane. To the needle are attached counterarms, which can be adjusted to a known angular relationship to the axis of the magnet. On one arm is a counterweight, which can be adjusted by varying its effective lever arm. Without the counterweight the assembly is balanced. The pivots of the magnet rest upon two parallel and horizontal agate edges. Centering devices and arrangements for raising and lowering the magnet are provided.

For determining the position of the magnet, a graduated circular scale is so placed that the magnet swings in its plane. It is graduated from zero at the top to 180 deg. on both halves. A mechanical finger and a small thermometer are also provided in the cylindrical brass case. A glass window covers one side of the case. The instrument is mounted upon a tripod with a four-screw leveling head. An accessory compass is also provided for orienting the instrument in the plane of the magnetic meridian. Figure 1 illustrates the important features of the instrument.

Apart from its excellent construction, the Hotchkiss Superdip instrument has the advantage that its sensitivity can be varied over a wide range by changing the angle of the counterarms, the limiting angle for maximum sensitivity being the complement of the angle of inclination. As the maximum sensitivity is seldom needed, a smaller angle is ordinarily used. Stearn states that at this maximum sensitivity one scale division is equal to a 3-gamma change in magnetic intensity (1 gamma equals 0.00001 gauss; the gauss is the c.g.s. unit of magnetic force and is equivalent to the force of a unit magnetic pole). Precise measurements with the instrument require corrections for temperature, diurnal variation, day-to-day variation, latitude, longitude, and for the base from which a magnetic survey is started. *Technical Publication* 370 of the A.I.M.E. gives many details of the use of this instrument.

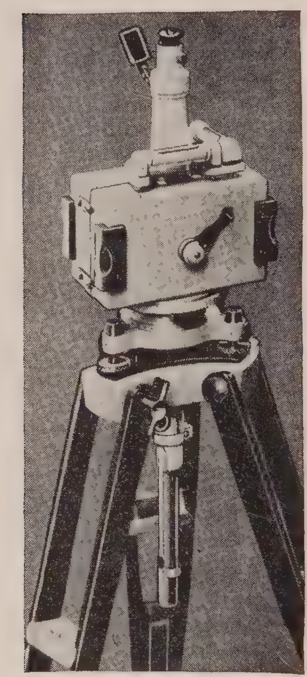

FIG. 2.—Askania vertical-field magnetometer.

The Askania vertical-field balance contains a magnet system consisting of two magnetized-steel lamellae of equal magnetic moment attached to the sides of a stainless-steel frame, which also carries three bronze adjusting weights. The weights at the ends of the center block are for adjusting the magnet system to any vertical intensity that may be necessary; the lower weight is for raising and lowering the center of gravity of the system so as to vary its sensitivity. A quartz knife-edge passes through the center of the system and is supported by quartz pieces of semicylindrical form. The mass distribution of the system is so arranged that the center of gravity is on the south polar side below the pivot and the needles are in a horizontal position. Devices for damping, for centering, and for placing the system in and out of action are provided. The whole is inclosed in a heavily insulated case. A small thermometer is placed close to the magnets. A mirror is attached to the upper side of the steel frame, and by a system of lenses the angular movement of the magnet system is measured upon a fixed scale, light for this purpose being reflected by a mirror attached to the outside of the small telescope. The instrument is shown in Fig. 2, and the system of mirrors and lenses in Fig. 3. Beneath the tripod top, in line with the principal axis, is a fixture for attaching the auxiliary magnets, which are carried in threaded sheaths. The arrangement is such that the vertical position of the auxiliary magnets can be varied and the distances from the centers of the auxiliary magnets to the quartz knife-edge can be read from a scale. The auxiliary magnets are used for extending the range of measurement and for determining the constants of the instrument. Corrections for temperature, diurnal, latitude, and base station are necessary for precise results. The instrument can measure magnetic anomalies of a low order. It has been extensively used in geological explorations in connection with petroleum and other minerals. C. A. Heiland, in *A.I.M.E. Technical Publication* 483, gives an interesting account of improvements in this instrument.

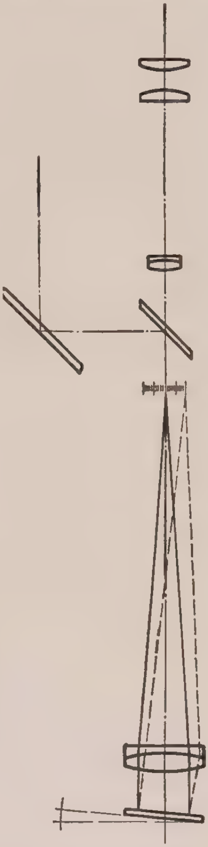

FIG. 3.—System of mirrors and lenses used in the Askania magnetometer.

Other types of magnetic variometers are available for the determination of the horizontal intensity of the earth's magnetic field. The Thompson-Thalen magnetometer, the Askania horizontal field balance, the Kohlrausch horizontal variometer, and the compass variometer are examples of instruments used for this purpose. A comprehensive survey includes accurate measurements of horizontal intensity. Relative measurements can be taken by the magnetic compass by observations at intervals along lines of known orientation.

A magnetic survey requires the systematic observation of vertical and horizontal intensities at regular intervals along a series of parallel lines equally spaced and covering the area under investigation. The mapping of the system, together with the observations taken at the various points, is the important preliminary to the interpretation of the results. Significant sections are constructed, upon which the horizontal and vertical intensities are plotted to vertical scale. In addition, isograms, or lines of equal magnetic intensities, are plotted upon the areal map. The final interpretation is worked out in conjunction with geological studies of the structure. The magnetic anomalies aid the geologist, particularly where the formations are masked by surface soil.

Gravimetric Methods.—Gravity is the force of attraction exerted by the earth in a direction towards its center on a body of unit mass. The gram is the unit mass, and the intensity of gravity at sea level is about 980 dynes. If the earth were built up of spherical shells each of uniform density and if it were a perfect sphere, the intensity of gravity would be equal at all points; but as the earth is a spheroid, the intensity of gravity is greatest at the poles and least at the equator. Likewise the rotation of the earth makes gravitational intensity less at the equator. Topographical irregularities also influence this factor, as does the variation in the density of the various masses that constitute the crust of the earth. It is this last fact that is taken advantage of in using a measure of the gravity intensity to determine the approximate position of structural masses that differ in their density. By geological-structural studies the nature of these masses can sometimes be inferred. Their outlines can be also approximately determined and their depth below the surface estimated more or less accurately. This is of importance to the geophysicist.

Variations in gravity attraction can be determined by accurate pendulum observations, but the Eötvös torsion balance has been generally adapted for the purpose and has been widely used in investigations of this kind. With it the gravity gradient, or the rate of the horizontal variation of the intensity of gravity, and a function of the curvature of the level surface can be determined. Gravity gradient is defined as the difference in gravity intensity per horizontal centimeter and generally refers to the maximum or total gradient. It is represented on maps by an arrow pointed in the direction of the increase of gravity and of a length proportional to the magnitude of the gradient. Barton states that the measurement of the gradient by the torsion balance is due to the fact that the warping of the vertical, that is to say the divergence of the level surfaces, produces at each weight of the balance a small horizontal component of gravity equal to the horizontal gradient of gravity. The balance

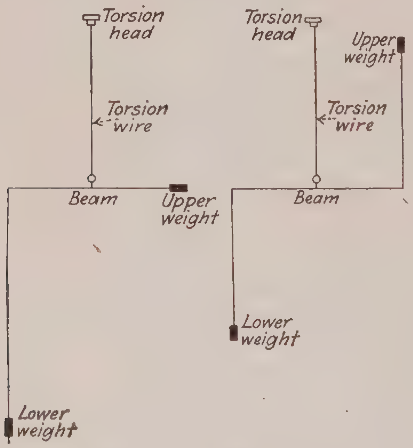

Fig. 4.—Element of Eötvös torsion balance.

consists of the elements shown in Fig. 4, a horizontal aluminum beam suspended by a wire and carrying two weights, one at the end of the beam and the other suspended by a wire at the other end. Both weights are equal. In the *z* arrangement one weight is carried above one end of the beam and the other below, as shown at the right in Fig. 4. The latter arrangement results in a more compact instrument. The balance is protected by an outer case, shown in Fig. 5, and contains as accessories a mirror system and a photographic recording device.

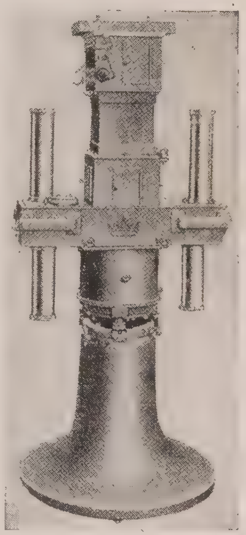
FIG. 5.—Torsion balance (Askania); height $47\frac{1}{4}$ in.; weight 93 lb.

The unit of measurement for the torsion balance is the Eötvös, or 1×10^{-6} dyne (per gram) per horizontal centimeter. Barton states the values of the gradient measured in the field range most commonly from 5 to 30 Eötvös, and the mean maximum gradient observed in his work was about 150 Eötvös. He states that the unit of measurement for the differential curvature should be in units of 1×10^{-12} radian per cm., as the significance of the magnitude lies in its indication of the difference of the warping of the level surface in different directions. The *R* that is actually used in practice, however, is gravity times that difference, and is measured in terms of 1×10^{-9} dyne per g. per cm. (Eötvös units). But as the reciprocal of the value of gravity in most places is within a few per cent of 1×10^{-3}, and as the field measurements of *R* are not accurate within a few per cent, the numerical value of *R* in terms of 1×10^{-9} dyne per g. per cm. (Eötvös units) is the same for most practical purposes as the numerical value of the differential curvature in terms of 1×10^{-12} radian per cm.

Determinations made by the torsion balance are the summation of all influences beneath the surface and near by. To get at the gravitational effect of earth masses, it is necessary to correct for topographic irregularities, whether elevations or depressions, and for the latitudinal variation in the earth's northward or southward gradient. These corrections are usually subtractive, leaving a remainder that is assumed to be the resultant of geologic anomalies. Barton divides these anomalies into five orders of magnitude: (1) Those due to masses of the size of large mountains or large geosynclines, (2) those due to the masses of the size of small mountain ranges, such as buried granite ridges, (3) those due to masses of the size of the average anticline, (4) those due to minor geologic structures, and (5) those due to small irregularities such as glacial boulders buried in the subsoil rather close to the instrument.

The torsion balance has been simplified and improved. Askania manufactures a new instrument of the photographic type, supported upon a light tripod. It weighs 55 lb. in the semiautomatic form and 85 lb. for completely automatic operation. By its use observation time has been reduced to 20 min. for a single station. Gravimeters of the spring type as well as improved pendulum instruments have been developed. The Hartley gravity balance is one of the former. Gravimeters are proving to be more convenient, rapid, and economical than the torsion balance and are displacing it. Heiland stated in 1939 that probably 100 to 125 gravimeter crews were in the field in the United States. Descriptions of gravimeters are given by him in *A.I.M.E. Technical Publication* 1049.

As gravimetric applications are rather narrowly limited to flat terranes, surveys require skilled interpretation. A gravity high or a low may be indicated upon the gravity map. A gravity high may indicate a rock mass of greater density than surrounding rocks. A gravity low in the Gulf Plain usually indicates the presence of a salt dome and a comprehensive survey by this method enables approximations of form, limits, depth, and thickness of cap rock to be made. Details of such a survey are given by D. C. Barton in *A.I.M.E. Technical Publication* 719. A similar survey upon a salt deposit in Nova Scotia is given by A. H. Miller and G. W. H. Norman in *A.I.M.E. Technical Publication* 737. Both of these papers are worth the attention of the student.

Seismic Earth-wave Method.—The velocity of earth waves produced by dynamite explosions varies appreciably in rock masses of different character. D. C. Barton gives the table of the velocity of longitudinal earth waves on page 41.

It is this difference in the velocity rate that is taken advantage of in differentiating underlying strata. Rieber explains the elementary principle in the following way: In Fig. 6, I, the earth section is assumed to consist of two horizontal layers that differ in their densities and therefore in the velocities with which the longitudinal earth wave is transmitted. If a series of successive explosions are made at the points *A*, *B*, *C*, *D*, *E*, and *F*, and the times for transmission are recorded at *R*, it will be found that the explosive waves from near-by points *A*, *B*, and *C* arrive at such time intervals that their velocities are the same. From the more distant points *E*, *F*, and *G* the velocities are greater. Thus the waves travel for a part of their course in the denser medium, which has a higher velocity rate. The effect of the explosion is radial from the point of explosion. Two waves may be assumed, one in the surface medium and one which is refracted downwards to the underlying stratum and at some point on the interface refracted upwards to the timing point R. Symmetry can be assumed as indicated in the figure. If time in relation to distance is plotted as shown in the lower figure, the velocity graph is a bent line, the first portion representing the near-by points and the second the more distant points. Distance divided by time is the velocity rate. Some point exists along the surface at which the time for the transmission of the surface wave coincides with that for the wave taking the longer path in the lower stratum. This point is indicated by the distance L_1 in Fig. 6, II. If it is assumed that the velocity in the upper medium is V_1, in the

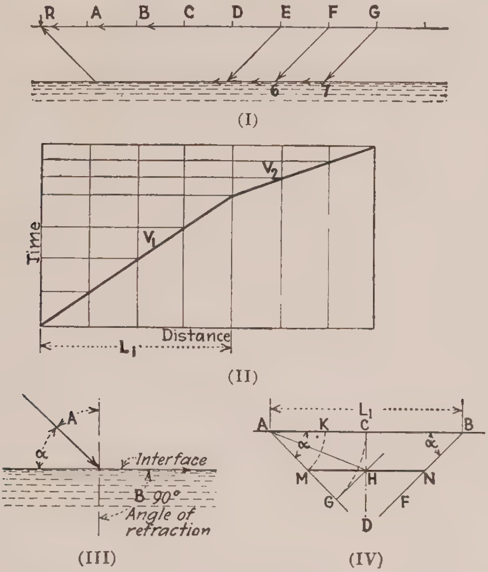

FIG. 6.—Relation between velocities in different media. (*After Rieber.*)

TABLE 7.—VELOCITY OF LONGITUDINAL EARTH WAVES

Rocks	Kilometers per second		
	Minimum velocity	Mean	Maximum
Crystalline schists	5.5	6.3	7.0
Plutonic rocks	2.1	5.3	8.0
Eruptive rocks	1.4	2.8	4.2
Sediments in general	1.2	3.6	5.9
Limestone	3.8	5.0	5.7
Archaean rocks	4.1	5.6	7.0
Paleozoic sediments	2.2	4.5	5.7
Mesozoic sediments	1.8	3.4	5.9
Tertiary sediments	1.2	2.1	3.0
Pleistocene sediments	1.4	2.2	3.6

lower medium V_2, the angle of incidence from the upper to the lower medium A and the angle of refraction within the lower medium B, then the relation follows: sin A is to sin B as V_1 is to V_2. If B becomes 90 deg., as is the case if the refracted wave skims the surface of the interface, then the proportion can be simplified by expression V_1/V_2 equals cos α where α is the complement of the angle of incidence as shown in Fig. 6, III. An inspection of Fig. 6, I, will indicate that the difference between the time observation for the explosion of G and that for the explosion at F will represent the time required to travel the distance 6–7 at the interface, and this divided by the difference in time will give V_2, or the velocity in the lower stratum. A simple graphical construction thus becomes possible. As constructed by Rieber, the angle α is determined from the ratio of the two velocities. A line AB equal to L_1 is laid off and lines AG and BF are drawn making angles α with AB. Draw a normal CD at the center between A and B. With a center at A, draw an arc AC intersecting the line AG at G. Erect a normal at G giving the intersection H with the normal CD. Draw the line MN through H parallel with AB. The position of the interface is MN and the depth is CH as shown in Fig. 6, IV. The proof is as follows: Travel time AB is equal to time $AMNB$ as AB equals L_1; time AC equals AMH; time AK equals time AM (equal distance in same medium); KC/t equals V_1; MH/t equals V_2; KC/MH equals cos α, but MG also equals KC and MG/MH equals cos α. The depth CH equals $\frac{1}{2}L_1$ tan $\frac{1}{2}\alpha$ from the construction.

Rieber, whose proof is given in the foregoing, points out that this simple computation is limited to the condition that the velocity in the lower stratum is higher than that in the upper and that the interface between strata is parallel to the surface. It is also obvious that it holds where the line of shots parallels the strike of an inclined bed, in which case he states that the depth obtained will represent the normal distance to the inclined bed rather than the vertical distance below the surface. The formula does not apply in cases where the underlying stratum either dips or rises in the line of the shots, nor where the velocity in either medium is not constant. However, he states that given a sufficient number of measurements in any area, it is possible to determine whether the lower stratum is dipping or rising in the line of shots or whether the velocity in either medium is varying, and to arrive at corrections to be applied for these conditions, so that final solutions may be brought to a fair degree of accuracy in most cases.

A line of shots received at a single station, or a single shot received by a series of instruments at intervals along a given line, is in the nature of a preliminary test of formations and conditions. Subsequent shots are fanned out from a given receiving station, each shot being at the end of a radius from this station, or where a number of receiving stations are used, a single shot at the focus sends its impulses to each receiving instrument. The distance between the point of explosion and the receiving station may vary but must be such that, with the weight of dynamite charge selected, the impulses received by the seismograph are of sufficient magnitude to register. A second fan-shaped series of stations may be imposed over the same area covered by the first, as shown in Fig. 7. The second receiving stations are at different points. Barton states that the shot length varies with the type of work and the situation. By the reflection method, the shot length is a function of the depth and the critical angle of reflection and ranges from 1.2 to 1.8 times the depth of the formation that it is desired to map or to avoid, and the respective velocities. By the refraction method, now seldom used, the shot length is 3 to 5 times the depth of the formation to be mapped and less than 3 to 5 times the depth of any lower high-speed formation that is to be avoided. He states that the shot length now commonly used in reconnaissance for salt domes along the coast of Texas and Louisiana ranges from 5 to 7 miles, slightly less along the Mississippi coast, and about 3½ to 5 miles in east Texas, but that where a formation having a velocity almost that of rock salt is present, the shot length is reduced to less than 3 to 5 times the depth of that formation in order to eliminate its effects.

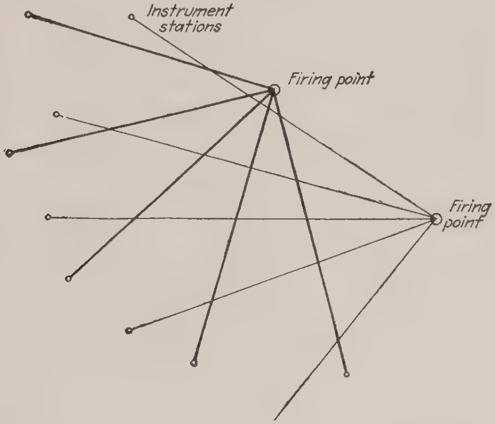

FIG. 7.—Fan-shaped nets established by two firing points and receiving instruments.

In recent years many improvements have been made in the instruments and technique of seismic work. Reflection exploration has displaced refraction methods. Geophones, amplifiers, recording cameras, and accessories are better and give greater control over field results. Field units for reflection work include a surveying truck with plane table outfit, a drilling truck with rotary drilling equipment and accessories, a shooting truck with portable magazine, and an instrument truck with its equipment in place. A crew of about 15 men is required.

Geophones react to earth waves resulting from a blast and generate a small current that is amplified by a vacuum tube such as is used in radio. The amplifier modifies or filters the signal by eliminating certain wave trains that might obscure the reflection record. The recording camera produces a record of each geophone signal upon photographic paper. The paper moves at the rate of 12 to 18 in. per sec., a light spot actuated by a galvanometer records a sinusoidal trace, and an electrical tuning fork produces a timing trace upon the same paper. The time of explosion of the blast is also recorded.

In reflection shooting, 6 to 12 geophones are set along a line at equal intervals 50 to 250 ft. apart. They are placed on either side of the shot position, or the shooting point may be at one end of the line, which may be 1,000 ft. or more in length. The times of arrival of the reflected waves are almost simultaneous at all the geophones for a given reflecting formation and thus can usually be identified. Several interfaces at different depths may be recorded by as many reflecting waves, each set of waves occurring in sequence, the deepest formation recording last. The photographic record thus indicates the time at which each reflecting surface is recorded on the several geophones. Elapsed time from the instant of shooting may be several seconds. Not all records are clear, so that experience is necessary for accurate determination of the position of the reflection wave upon the chart. The position and depth of the shot, as well as the position and elevation of each geophone, are given by the survey.

Practice varies in the position of the line and the placing of geophones. Each field develops its peculiar technique. Continuous profiling locates two shot points, one at each end of the line, the first shot point having its last geophone at the position of the second shot point; after the record for the first shot point is completed, the last geophone of the first shot is moved to the first point and the second shot is recorded. A reflecting point common to both shots is thus obtained.

In reflection shooting, many corrections are necessary before accurate time intervals are found. In addition, experiments are necessary to find out wave velocities in different formations. These velocities are used to compute average velocities of reflection waves. In *A.I.M.E. Technical Publication* 833, S. J. Pirson gives detailed examples of computations necessary to determine the space position of the reflection points.

Comparatively shallow holes and small charges of explosive characterize reflection as compared with refraction shooting, where charges from 40 to 250 lb. may be required. Passing through soil, the hole bottoms in the unweathered zone some 5 to 40 ft. down. Permanent ground-water level is usually a satisfactory depth. Charges of explosive range from a few pounds up to 15 to 30 lb., which is considered large.

Reflection seismography has for its principal objective the determination of the position, dip, and depth of a given formation or formations that may have significance in the search for oil. Brandt states that it has carried structural petroleum exploration down to 15,000 ft., that it is applicable to extensive, gently dipping formations, and that strata of 50 ft. can be observed at a depth of 10,000 ft.

Electrical Methods.—Sulphide ores, as distinguished from oxidized ores, have an electrical conductivity many times that of accessory minerals and rock formations in which they may occur. The one notable exception is sphalerite, of which most varieties appear to have a very low electrical conductivity. A metalliferous deposit below the surface, in which chemical changes due to circulating ground waters are taking place, may be the source of comparatively feeble earth currents. The detection of these earth currents or the determination of the locus of superior conductivity is the objective of electrical methods of prospecting. These methods have been developed in great variety and have obtained sufficient success to warrant the interest that is taken in the subject. However, many electrical surveys have failed to respond to the test of the diamond drill or to underground exploration, and as a consequence widely differing opinions as to their efficiency are held.

The surface potential method was first introduced by Conrad Schlumberger. In principle it is based upon the fact that an electrical current passing through the earth between two points is subject to distortion by the presence of an underground conductor, and that these distortions in the field can be detected by determining equipotential lines upon the surface. If the conductor is at too great a depth or is of small dimensions, it will produce no detectable surface effects. In contrast to the foregoing, the self-potential method does not use an artificial current but determines the surface effects of the earth currents generated by the ore mass below. In both variants of the potential methods, nonpolarizing electrodes are used to establish the connection with the ground. The construction of such an electrode is shown in Fig. 8. In the surface potential method, lines of equipotential are determined by the use of two electrodes connected by an insulated cable in the circuit of which are a two- or three-valve amplifier and a telephone, where alternating current is used. Where a direct current is used, a sensitive galvanometer is

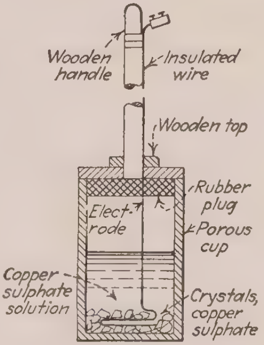

FIG. 8.—Nonpolarizable electrode. (*After Schlumberger.*)

placed in the circuit of the electrodes. In the self-potential method, a sensitive portable galvanometer determines the direction and magnitude of the earth current and a microammeter the current flow.

In tracing out equipotential lines where an a-c field is used, one electrode is fixed and the other is used to search for a position where the potential is the same, which is indicated by silence of the telephone receiver. The first electrode is moved to the position occupied by the second, and the exploring electrode is used to find another point on the equipotential curve. In this way a number of equipotential lines are traced out upon the surface. The positions are recorded upon a map. Figure 9 illustrates such a map. The dotted lines indicate current flow and the solid lines equipotential curves. A conductor is indicated by the crowding of the curves about its apparent position as shown in the figure.

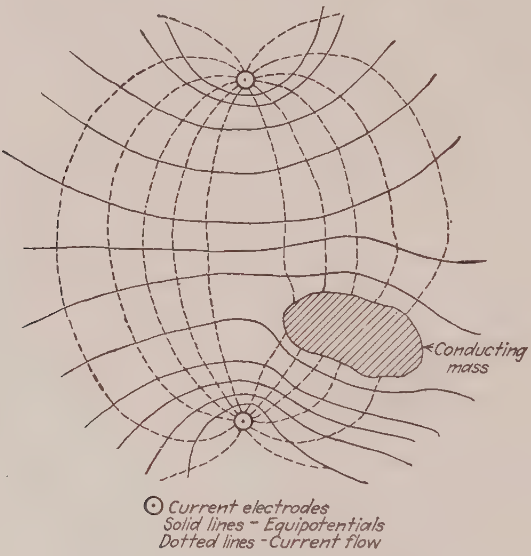

FIG. 9.—Equipotential lines; a-c field.

Tracing of equipotential lines in the self-potential method is similar. Zero current in the galvanometer indicates that both the electrodes are on the same equipotential curve. Upon the surface above the conductor, the equipotential lines form a series of concentric ellipses as shown in Fig. 10. Eve and Keys suggest the following method: A course is selected and observations are taken at 100-ft. intervals. Holes are scooped out and watered for the reception of the nonpolarizable electrodes. The current flow is measured by a microammeter, and the direction of current flow is noted as each new point is occupied by the forward electrode. They state that the resistance of each electrode when placed in a well-watered hole is about 500 ohms, so that the total resistance of the circuit is about 1,000 ohms, a microampere corresponding to a potential difference of a millivolt. On the completion of readings, distances and plus or minus value of the current are plotted. In traversing a vein, the current values gradually increase to a maximum over the vein and then abruptly reverse to an approximately equal value as the center of the vein is passed, after which the readings gradually diminish and resume their minor fluctuations as the vein is passed.

Lundberg states that terrestrial currents are generally too weak and unsteady to be useful for the purpose of electrical prospecting. Alternating current is preferable to direct current because it admits of amplification and avoids the difficulties caused by chemical polarization. The alternating current is impressed through two parallel conductors, which are brought into contact with the ground at close intervals by means of iron pins 1 or 2 ft. in length. Single-phase alternating current of audio frequency 50 to 10,000 cycles is used. Figure 11 represents the conditions of current flow in the presence of a conductor. The investigation of this electrical field is restricted to the surface and is similar to what has already been described. Equipotential lines are at right angles to the current flow, and their shape indicates the distribution of the current underground. Certain general conditions are given by Lundberg: In barren ground the current flows uniformly, is of normal density, and the fall of potential constant but fairly large, owing to the low conductivity of the ground. In the ore mass the current density is much higher, and on account of the superior conductivity of the ore the potential drop becomes very small when compared to that of barren ground. The converging of the current to the ore body causes increased density in the wall rock just before the current passes into the ore body proper. The density will be increased above an ore body and on the sides. Observations of potential taken at equal intervals of 5 to 10 ft. apart along a line transverse to the length of the ore body will show pronounced maxima above the boundaries of the ore body. Equipotential lines can be traced by means of two iron electrodes connected to a telephone receiver,

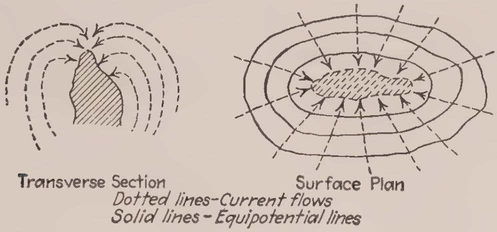

FIG. 10.—Self-potential method.

points of equal potential being indicated by the absence of sound in the receiver. The points are marked out upon the ground and then recorded upon a map. They are shown as curves in Fig. 11.

An interesting application of electronic equipment has been made by G. R. Fisher of Palo Alto, Calif. The prospecting instrument consists of a transmitter and a receiver contained within two parallel wooden bars of such length that a distance of about 6 ft. separates the two. The space between the bars is occupied by the user. Shoulder straps support the instrument, which weighs about 26 lb. The transmitter contains a battery and transmitting loop, the frequency being about 125 kilocycles. The receiver has a receiving loop and amplifier as well as controls and an instrument for indicating unbalance and its degree. Under normal conditions with the instrument level the indication is zero. When the instrument is carried over an underground conductor a secondary field results and distortion or unbalance is indicated. The range of the finder is limited to an area of about 4 ft. on either side and a depth of some 25 ft. Within its range it will indicate buried metals and minerals that are conductors. Probably the principal use to which the M scope has been applied is the locating of pipe lines by many utility and water companies. Fisher has constructed modifications of the instrument which show greater depth range and which may be useful for ore prospecting.

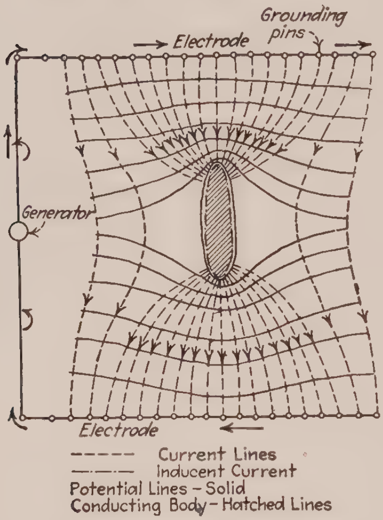

FIG. 11.—Single-phase a-c field in presence of a conductor. (*Lundberg.*)

Resistivity.—Measurement of earth resistivity has found widespread application in near-the-surface exploration. It is an aid in unraveling geologic structure such as contacts, faults, bedrock positions, water zones, and the presence of conductive masses where they are surrounded by highly resistive rock masses. Its findings are not always accurate. As in other geophysical methods, geological study is essential as an aid to interpretation of resistivity measurements.

Measurement methods stem from the work of F. C. Wenner. Figure 11*a* shows an arrangement of instruments and electrodes for the determination of earth resistivity. Two primary electrodes supply current. They are iron rods driven a foot or more into the ground and receive current from a d-c generator or battery. A potentiometer, connected to two secondary electrodes P_1 and P_2, is used to measure the voltage between the electrodes. A commutator K_1 reverses the current flow at the rate of 30 reversals per second and thus screens off stray currents; a second commutator K_2 serves to rectify the alternating potential between P_1 and P_2. By adjustment of the resistances R_1 and R_2 for a null position of the galvanometer, the current A and the voltage at V_2 may be read. The electrodes are in line and spaced at equal distances a. Specific resistance of the earth is given by the following equations, after G. F. Tagg:

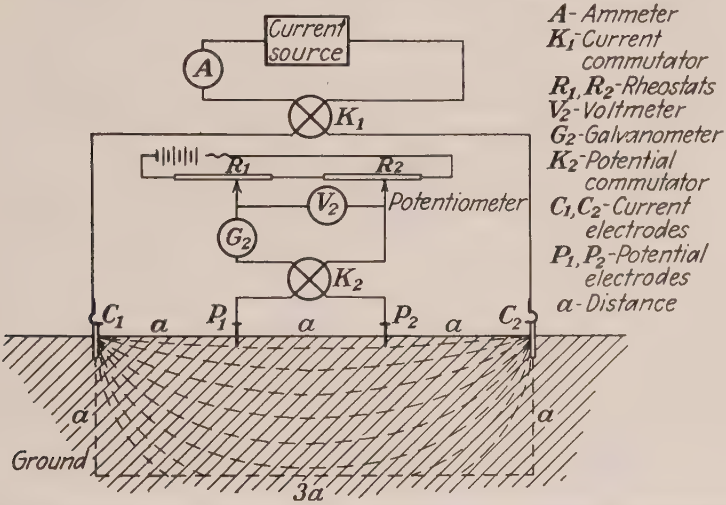

FIG. 11a.—Arrangement of instrument and electrodes for determination of earth resistivity. (*A.I.M.E.*)

$$\rho = 2\pi a \frac{V}{I}; \qquad \frac{V}{I} = R \text{ ohms}; \qquad \rho = 2\pi a R$$

where V = volts
I = amperes

If a is in inches, ρ will be in ohms per inch cube, or ohm-inches; if a is in centimeters, ρ will be in ohms per centimeter cube, or ohm-centimeters.

The position of the station, the electrode separation, and the azimuth of the line of electrodes may be varied. Hubbert states that the apparent specific resistivity is measured as an empirical function of any or all of the variables. He gives several techniques: A longitudinal traverse profile signifies the maintenance of the line of electrodes parallel to the traverse line; a transverse traverse profile requires the line of electrodes to be at right angles to the traverse line at successive stations; a circular profile indicates the rotation of the line of electrodes about a station at regular angular intervals; a depth profile results from successive increases in the value of a, the electrode separation, with station position and azimuth unchanged. A resistivity survey includes the establishment of a number of carefully surveyed stations or stations tied into survey marks, at which points the measurements are made.

A large number of resistivity measurements may be required in exploring a given area. In depth determination the results of a series of measurements with the center of the electrode system at one point and successive increases in the electrode distance are graphically plotted against the electrode interval a. A sudden change of curvature is assumed to indicate that the earth mass causing this sudden change lies at a depth equal to the electrode interval at which the change took place. Such empirical methods of interpretation are based upon the assumption that all the electrode positions are in one plane and that the intervening earth mass is homogeneous, which may be so in many cases.

Two papers are suggested to the student: Results of Earth Resistivity Survey, by M. K. Hubbert, *A.I.M.E. Technical Publication* 463, and *A.I.M.E. Technical Publication* 683. Both give examples of resistivity surveys. F. C. Wenner's paper, A Method of Measuring Earth Resistivity, is published as *U.S. Bureau of Standards, Bulletin* 12, 1916.

Miscellaneous Methods.—Many geothermal data and underground temperatures have been recorded. Temperatures may be accurately measured by thermocouples designed for this service, although the mercury thermometer is more often employed. Temperature surveys of boreholes may result in something of value in mineral search. The significance of such temperature determinations is more or less unknown at present, but research may develop important correlations with structures.

Radioactivity has importance in determining the age of rocks and has been of use in correlation of sedimentary strata. It has also played a part in the search for radioactive minerals and their deposits.

It is rumored that radar is to play a part in mineral search, but no information is available at present. It is still a military secret.

The examination and analysis of the gases found in soils are of importance in the search for oil and gas deposits. Rock analyses are sometimes used in the correlation of geological formations. Petrographic and spectrographic methods are used more and more frequently. Microanalysis is assuming greater importance in the examination of mineral fragments.

Aerial reconnaissance and photography are important in many mineral areas as an aid in the recognition of faults, formations, and structural features, as well as for surveying and mapping. C. H. White has employed color photography in his study of outcrop areas and has made use of color analysis for studying of rock specimens from significant alteration areas. C. A. Dobbel has made use of color photography for diamond-drill core records.

Future of Prospecting.—Long-tried simple methods of prospecting will continue to be used and a continued dependence upon surface showings is inevitable. Chance will likewise play a part in discovery. The trench, the borehole, and exploratory workings are the mining engineers' probes. Though the amount of geological and mineral lore is very large, redistillation, condensation, and obsolescence are decreasing the bulk of significant material. What remains is still too much, however, for a single individual to know. Specialization is the answer in the fields of metals, industrial minerals, coal, gas, and petroleum, and these may be further divided. With respect to methods, specialists have been trained in mineralogy, chemical and spectrographic analysis, petrography, paleontology, geology, and the numerous divisions of geophysics. The mining engineer can not hope to compete with these specialists, but he should avail himself of their professional experience in the solution of his problems. The probabilities of success in mineral discovery are enhanced if the talents of specialists can be teamed at not too great a cost. It is also necessary that the mining engineer have a general knowledge of methods.

The principal qualification of the technical explorer, according to A. Locke and P. Billingsley, is his belief that improvement can be made, and his equipment should include all possible devices that may be available from the world of science and industry. It is his special purpose to know everything that might find application. He should take the records of petrography, stratigraphy, and structure, and the group of data gathered by geophysical testing all into account and then reckon the probability of finding the hidden ore.

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CHAPTER III

BORING

Purposes of Boreholes.—Boring and drilling, signifying the penetration of alluvial material, rock, or formation by holes of relatively small diameter, are important in most mining operations. Boreholes are sunk for prospecting and formation testing; for outlining and sampling alluvial deposits, beds of salines, and ore bodies; for extraction of gas, petroleum, sulphur salt, and other salines; for ventilation and drainage; for pipes, electrical conduits, and ropeways; for sand filling and culm flushing; and for fire fighting. In addition, relatively shallow holes are sunk for foundation testing for bridges, dams, and building sites.

Methods and Limitations.—Hand methods are relatively unimportant but useful. They comprise various adaptations of the posthole auger and churn drill (1). The Empire drill (2), used in preliminary placer prospecting, makes use of both the churn drill and the auger. A well drill of the churn-drill type (3), power driven, is in general use for sampling placer deposits and near-surface ore bodies of lead, zinc, and copper. For oil and gas wells, the hydraulic rotary drill (4), first used in petroleum operations in 1901, has practically superseded the churn drill. The jetting drill (5) is in restricted use. Two other methods of importance, both rotary, are the diamond drill (6) and the chilled-shot drill (7).

Most boreholes are drilled vertically. With the exception of the diamond drill, all methods are restricted to vertical holes, although the chilled-shot drill can be used for slant holes at moderate angles from the vertical. Holes are never actually vertical except near the surface; all have been found to deviate to a greater or less extent with depth. The diamond drill can be used to drill in all directions. The hydraulic rotary is sometimes used to drill slant holes by whipstocking, but such holes are started vertically.

In all methods, except the chilled-shot drill and the diamond drill, the broken-up material over the area of the bore forms a sludge that is either continuously removed in mud-laden fluid, as in the hydraulic rotary method, or intermittently removed by a bailer, as in the churn-drill method. In diamond and chilled-shot drilling a core as well as a sludge is produced, the latter being washed out by a stream of water introduced through the drill rods.

Boring may be accomplished by a rotary cutter or auger; by a chisel-shaped bit raised by a rod or rope and allowed to fall, being rotated through a small angle at each stroke; or, in core drilling, by rotary cutters with or without hard abrasives set in the cutters, as in the diamond drill, by feeding chilled shot or steel fragments into the space between the wall of the bore and the core. In rotary methods, the weight of the drill rods and cutting tools, the speed of rotation, the kind of cutting tool, and the nature of the rock determine the rate of drilling; in churn drilling the weight of the string of tools, the length of stroke, the number of strokes per minute, and the nature of the rock determine drilling speed.

The unstable nature of soil and some sedimentaries necessitates the use of steel casing to support the walls of the bore, especially near the surface. Such a casing is driven from the surface. In unstable formations it follows the drilling operations closely or sometimes even precedes them.

The diameter and depth of a borehole are restricted by purposes for which the borehole is to be used and by practical limitations. Shafts of large diameter have been sunk by boring methods, but these do not concern us here. Hand methods of boring are applied to holes ranging from 2 to 6 in. in diameter and to depths under 100 ft. All other drilling presupposes power, gasoline engine, diesel engine, electric motor, or steam engine.

The deepest borehole as of Dec. 29, 1944, is Standard Oil Co. of California KCL 20-13 in Kern County, which was discontinued at a depth of 16,346 ft. without penetrating to commercial production. A 3½-in. drill pipe was used in the deepest drilling. Other deep wells are the Phillips Petroleum Co. Ada Price No. 1, Pecos County, Texas, completed in 1944 at a depth of 15,278 ft. and the Continental Oil Co. KCL A-2, Wasco, Calif., completed in April, 1935, at a depth of 15,004 ft. The deepest churn-drill hole is 7,248 ft.; the deepest diamond-drill hole was drilled upon the Rand, South Africa, to a depth of 10,718 ft. in 1938, being finished at a diameter of 2⅙ in.; in 1940 the second deepest hole, also on the Rand, was completed to a depth of 8,702 ft. with a bottom diameter of 3 in. It was drilled with a Longyear No. 4 drill. These are all exceptional records.

Core drills, except in very soft, fragile, or broken material, give an excellent record of the rock penetrated. The orientation or position of the core can be approximated, or more exactly determined where boreholes are surveyed. Some 100 per cent of core may be recovered in firm rocks. Experienced drill operators are necessary where a high percentage of core must be recovered in difficult formations. Except in shallow auger boring all boring methods yield a sludge from which a study of small particles must be made with a microscope, by panning, or by some other method. In placer prospecting the sludge is measured and washed to recover gold and accessory heavy minerals;

in other cases the sludge is thoroughly mixed and samples are cut out by various types of sample splitters. The hydraulic rotary delivers its cuttings in the mud fluid used in drilling, and the separation of the cuttings by screening is not satisfactory. Where accurate formation testing is necessary a coring attachment is used with the rotary, and significant cores are obtained at important strata.

Description of Methods.—(1) Hand augers are used for testing shallow alluvial deposits. Four types of augers are shown in Fig. 12. Posthole augers *A* and *B* are quite satisfactory in sand, earth, and similar material. *C* and *D* are used with the Empire drill. *D* is used for clay and *C* for ordinary material. Extension rods are constructed of ¾- or 1-in. pipe or, as with the Empire drill, solid square rods 1 in. in section can be used. For turning, a tee fitting and pipe handles are used with the pipe and a special type of handle with the square rods. The cycle of operations needs no description. For relatively deep holes a derrick 15 to 25 ft. in height, equipped with lifting tackle, is necessary. Light riveted casing of the "stovepipe" type or simple drive pipe is used in sand and loose material. The casing is driven a short distance; then the auger is inserted and the sand is removed to a depth of a foot or more, after which the process is repeated.

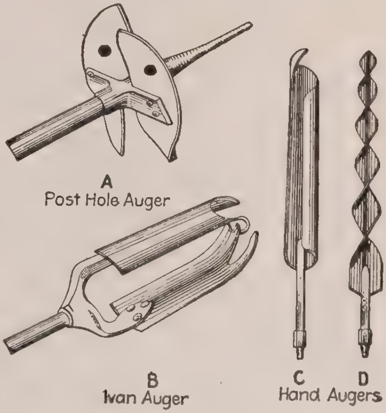


FIG. 12.—Hand augers.

The rate of drilling depends on the diameter of the bore, the nature of the material, the tools, and the experience of the men. For example, two men working 11 hr. and three men 4 hr. drilled a 2-in. hole 40 ft. deep in clay shale and wash iron ore; two men put down a 2-in. hole 18 ft. in clay, sand, and sandstone in 5 hr.; two men working 5 hr. and three men 25 hr. put down a 63-ft. hole 2 in. in diameter in clay, iron ore, and sandstone.[1]

2. Empire Drill.—The Empire drill (Fig. 13) is designed for prospecting alluvials. The drill consists of a flush-joint casing, on the lower end of which a toothed cutting shoe is placed and on the other end a light steel platform upon which the drillers stand. The platform is provided with an attachment in which a sweep can be inserted and the casing and platform rotated by manpower or horsepower. Rotation keeps the casing loose and free. In soft ground the drilling tools consist of augers; for firm ground or rock, bits similar in construction to those used in

[1] *Min. Sci. Press*, vol. 74, p. 452; *Trans. A.I.M.E.*, vol. 27, p. 123.

well-drilling outfits. They are attached to solid drill rods (5-ft. sections), which are raised and allowed to fall. Three or four men are required for the ordinary outfit. Cuttings are removed by sand pumps, of which two kinds are used. Both are provided with ball valves for the retention of the cuttings and are used with the drill rods. The "drilling pump" is equipped with a cutting shoe so that cutting and removal of

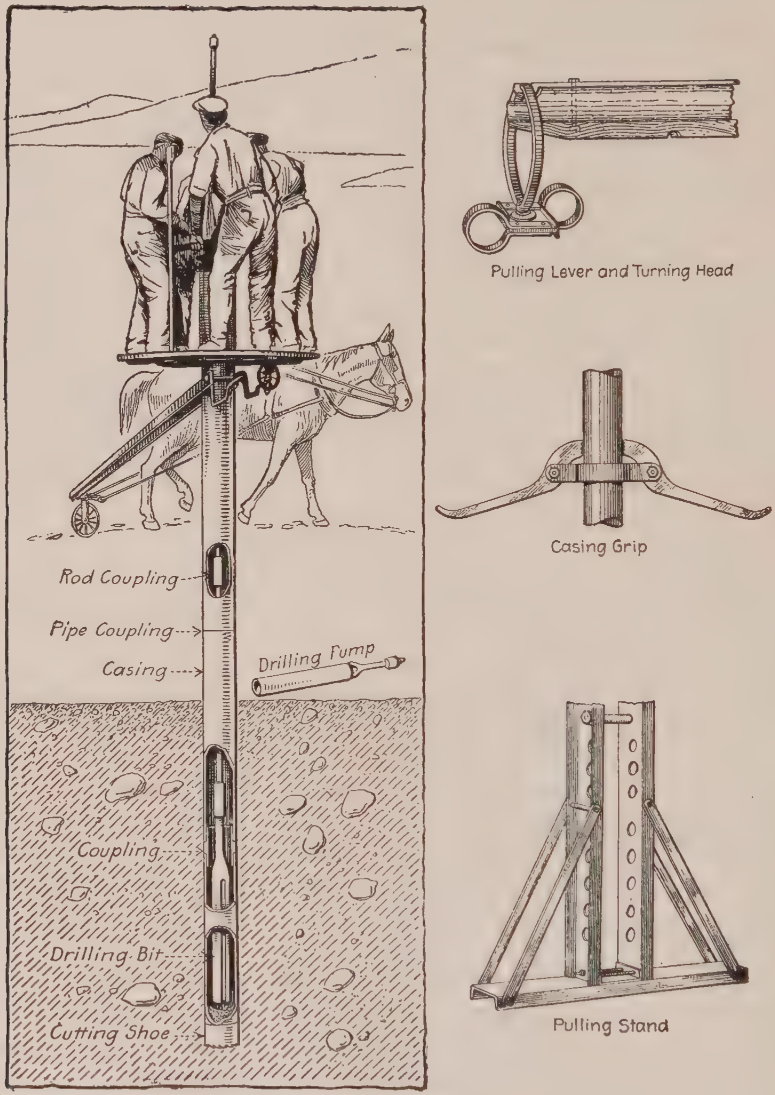


Fig. 13.—Empire prospecting drill.

cuttings can be simultaneously effected. As the hole is sunk, 5-ft. lengths of casing are added, and when the desired depth is reached the casing is pulled by attaching a special pulling head and using a stand and lever. Figure 13 illustrates the method of operation and the arrangement for pulling the casing. The weight of a 4-in outfit, equipped with 25 ft. of 4-in. casing, is 1,500 lb.; 50 ft., 2,000 lb.; 75 ft., 2,500 lb., and 100 ft., 3,000 lb. The weight limit of each piece is 75 lb. A smaller drill for a casing 2.5 in. in diameter is also manufactured.

3. Rope or Churn Drill.—Modern practice has made many mechanical improvements in this method, which is of great antiquity. The development of special metals and alloys has resulted in improved design of drilling rigs, of which many specialized types are available. Two are widely used in mining operations: One is the so-called "prospecting drill," the other a much heavier type also used for prospecting but more generally used for blasthole drilling in open pits and quarries. Alluvial deposits, sedimentary formations, altered monzonites, and moderately hard rocks can be bored with the churn drill. It is not adaptable to or at best is too slow for the hard, compact, and tough metamorphics and crystalline rocks.

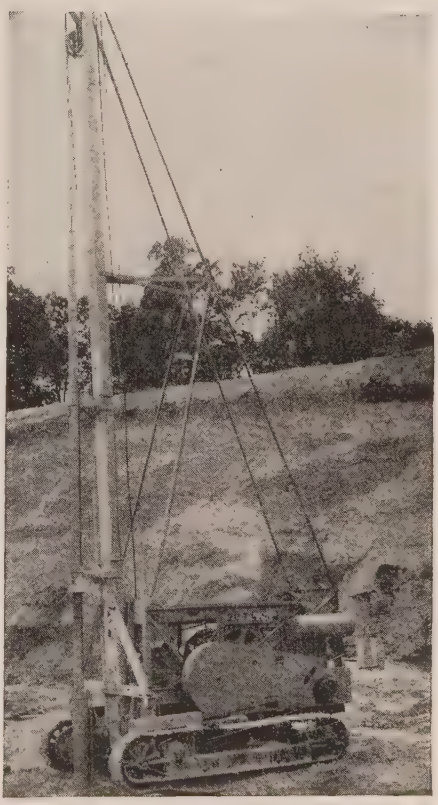


Fig. 14.—Prospecting churn drill. (*Bucyrus-Erie Co.*)

The cutting tool consists of a steel bit weighing from 150 to 200 lb., attached to a drill stem 12 ft. in length (in the lighter drill rigs) and 4.25 in. in diameter, and to a wire-rope swivel socket. The swivel socket automatically turns the tools at each stroke. Older types of tools included jars in the string of tools, but these have been superseded by sand-line-operated jar bumpers. Bit, stem, and socket are connected by standard screw joints of the tapered type used in oil-well tools. The string of tools is suspended by a left-laid $\frac{5}{8}$-in. wire-rope drilling cable. The sand line is a regular-lay wire rope, $\frac{3}{8}$ in. in diameter. The placer prospecting rig is equipped with two kinds of bits, the placer bit and a heavier rock-type bit.

Tool equipment weighs about 1,861 lb., and the maximum weight of the string of tools is 1,000 lb. The operating mechanism consists in part of a 22-hp. four-cylinder gasoline engine driving a jackshaft pulley through belt and belt tightener; the jackshaft carries a clutch and pinion to drive the spudding gear, a clutch and pinion to drive the "bull-reel" gear (drilling-rope drum), and a friction pulley to drive the sandline reel pulley. A worm-gear feed, operated by a handwheel, controls the movement of the bull reel during drilling. The normal speed of the spudding beam is 48 strokes per minute. Three lengths of stroke are available by changing the position of the crankpin on the spudding gear: Maximum 30 in., intermediate 24 in., and minimum 18 in. Figure 14 is a general view of a prospecting drill.

Two kinds of bailers are included in the equipment of prospecting drills. One is the vacuum type, 4½ in. by 8 ft., and the other a flat-valve or dart-valve type, 4¾ in. by 10 ft. Stuck drills are worked loose by a jar bump, which fits around the drilling cable and is raised and allowed to fall upon the rope socket by means of the sandline. The jar bump measures 2½ in. by 12 ft. Drive clamps, fitted with slip bushings to enable quick attachment to the drill stem, are necessary equipment for driving casing. Driveheads fitting over the outside of the casing pipe with tapered threads to fit the pipe prevent damage to the casing while it is being driven. Drive shoes with sharp-angled cutting edges are provided for attachment to the bottom section of the casing line, which is extra-heavy drive pipe 6 in. in diameter (28.57 lb. per ft.). Other equipment consists of a tool-sharpening kit, square tool wrenches, chain tongs, bar-and-chain tool tightener, and a casing-puller jar attached to the drilling stem and provided with a knocking head that screws into the pipe coupling. The jar is operated by the spudding beam. Fishing tools of various kinds are sometimes included in the equipment but are not often required for shallow-hole work. For holes deeper than 100 ft., fishing-tool equipment should be included. Figure 15 shows the tool equipment.

The rate of drilling depends upon many factors, such as the total time for moving, setting up the drills, actual drilling and casing, pulling the casing and preparing to move, and the footage drilled, as well as upon the weather, the nature of the ground, the experience of the drill crew, delays, and the availability of supplies. Rates for placer prospecting range from 10 to 25 ft. per shift; for soft ground, 20 to 50 ft. In frozen ground, where casing is sometimes not used, drilling rates are higher. In copper-ore prospecting, the average for holes 400 to 600 ft. in depth was 21 ft. per 12-hr. shift. Moving the rig, delays, repairs, and lowering

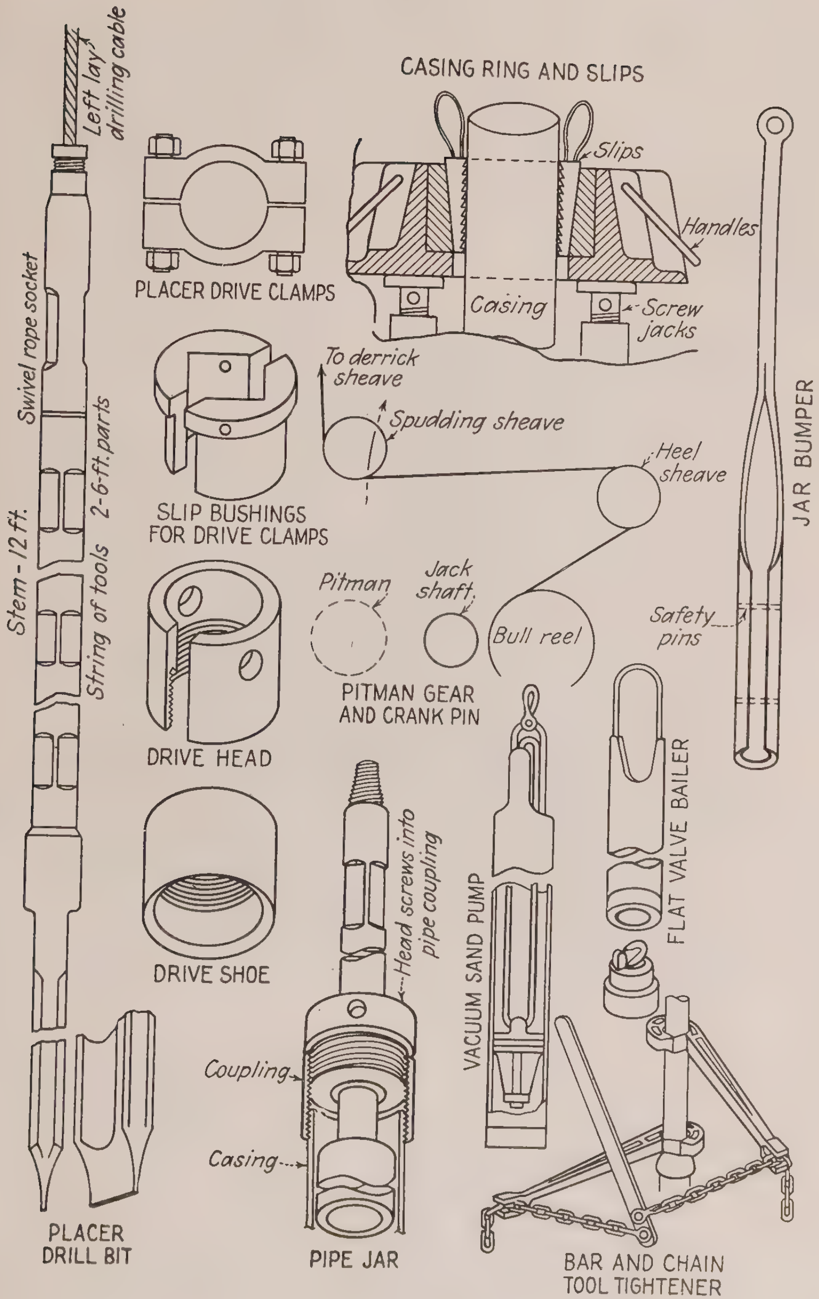


Fig. 15.—Churn-drilling tools and sketch showing spudding gear of churn drill.

and pulling the casing may consume 40 per cent of the total time, leaving some 60 per cent for actual drilling. Costs vary greatly. At Nome, Alaska, Rickard reported 93 cents per foot as the cost of drilling frozen ground without casing and \$1.50 to \$3.00 per foot with casing; Notman gives the average cost of drilling 60,000 ft. in Arizona (copper-ore prospecting with holes 400 to 600 ft. deep) as \$1.97 per foot. Present-day costs would be from 1.5 to 2 times the foregoing.

In placer drilling, large boulders are sometimes encountered and cause difficulty. When square with the hole they are drilled through, but when the top surface is on a slant, drilling may cause the bit to angle off, thus fitchering the hole, sometimes sticking the bit, and causing the casing to bend. In drilling through boulders, sand is first removed, and sometimes thick clay is thrown into the hole to form a sludge that will carry the sand in suspension and enable the bit to strike the boulder. The smaller boulders are smashed. Slant boulders are blasted by the following method: A charge of explosive with electric detonator is placed in a short section of stovepipe closed at its lower end by a plate of wood, the casing is pulled up about 3 ft., the charge is lowered to the bottom, and the blast is set off from a safe distance. After blasting, the

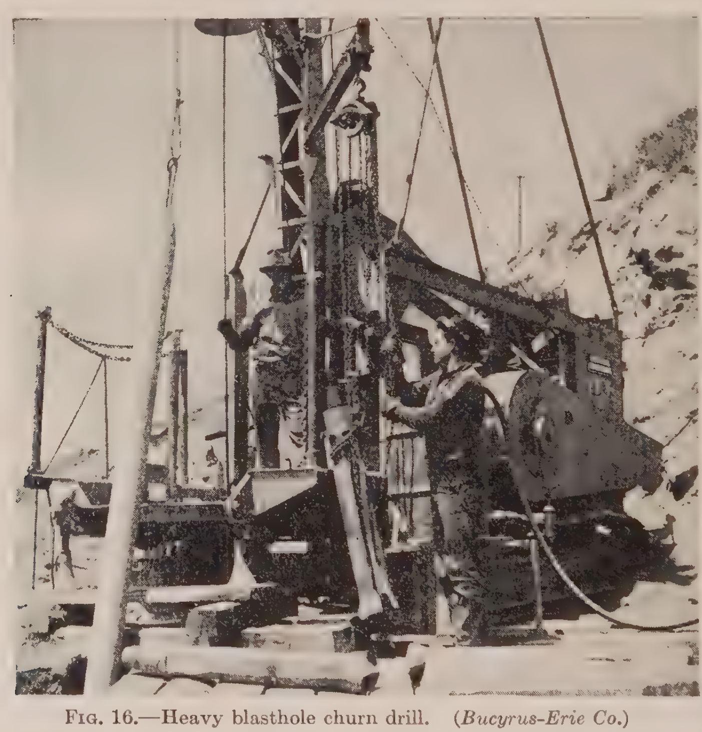


FIG. 16.—Heavy blasthole churn drill. (*Bucyrus-Erie Co.*)

hole is thoroughly cleaned out to about 6 in. below the blast before regular drilling is started.

Boring for sampling purposes, other than placer, involves deeper drilling and heavier drilling rigs, but otherwise the principles are much the same as in placer prospecting. Casing is used to ensure a uniform bore. As the formations are more homogeneous than the sands, gravels, and alluvium encountered in placer ground, drilling is more uniform; casing is harder to drive and drilling ahead of casing is general; underreamers may sometimes be necessary; casing may be harder to retrieve; and tool accidents may be more frequent. Boring for blasting purposes seldom requires casing except at the collar of the drill hole, and the work is on the whole simpler than where sampling is necessary. Holes are from 35 to 150 ft. in depth. Figure 16 illustrates a heavy drill for blasthole work.

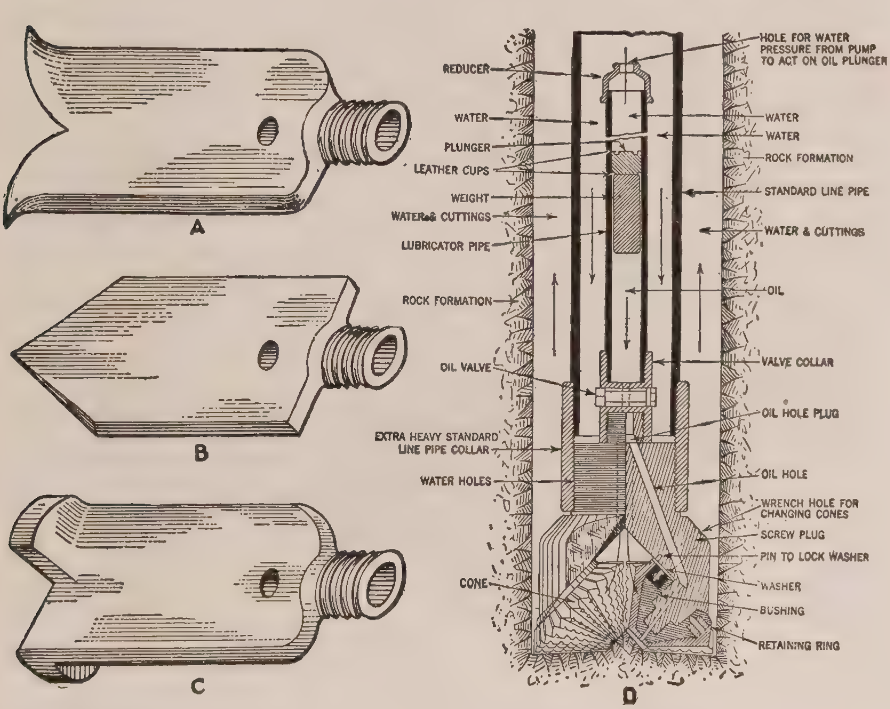


FIG. 17.—Rotary-drill bits.

In both placer and other sampling operations, two records are kept. One is the drill operating record, and the other is the formation and sampling record. The former is a time log involving moving up, preparation, drilling, trouble, drawing casing, dismantling, and delays. The latter log gives vertical position of samples, sampling length, formation changes, observations on mineralization and details affecting handling and marking samples. In boreholes for blasting purposes, the log of drilling operations with notes on formation changes is all that may be required.

4. Hydraulic Rotary.—In its modern application the hydraulic rotary drill consists of a sectionalized steel tubing joined by tapered screw joints, the upper end being attached to a hollow bar of square cross section, the kelly, and the lower to a cutting bit such as the fishtail (*A*, Fig. 17), the diamond point (*B*), or the drag bit (*C*), which is sometimes used with chilled shot in drilling hard formations. Various designs of the Sharp and Hughes cone bit (*D*) have found wide application for drilling hard rocks.

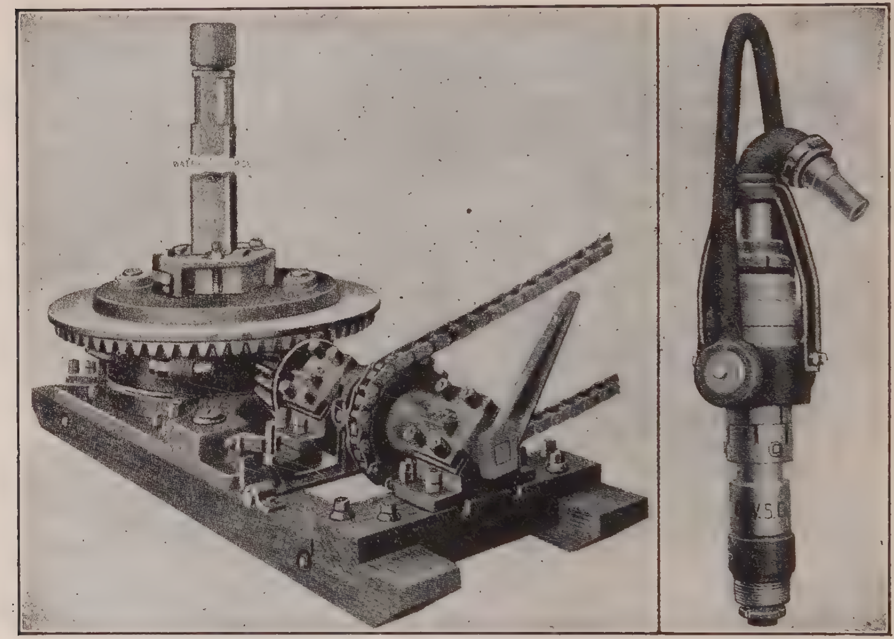


Fig. 18.—Rotary-drill turntable and swivel connection.

Figure 18 illustrates an early form of turntable, which rotates the kelly bar and allows it to be fed down as the bore deepens. A swivel connection is attached to the kelly bar at the top, and the whole drill string is suspended by wire ropes and blocks from the crown block of the derrick. Mud fluid is circulated by pumps, which deliver to a hose connection with the swivel joint, Fig. 18. As the bore is deepened, successive lengths of drill pipe are added. The mud fluid supports the sides of the bore and returns to a sump along with the cuttings. The weight of the drill string plays an important part in the boring and may be diminished by the tackle suspension. Table speeds range from 26 to 143 r.p.m. Rates of boring may be as high as 200 ft. per day. Stationary and portable rigs are in use. Steam, diesel, or electric power may be used. This method is utilized almost exclusively in boring petroleum and gas wells, though a few applications have been made in blasthole drillings.

5. Jetting Method.—The jetting drill is used to sink prospecting boreholes through surface formations such as gravel, soil, and glacial drift, as well as in iron ores. It consists of a sectionalized hollow steel rod on one end of which is a chopping bit, chisel shaped, and on the other a swivel hose connection and bail for raising and lowering by wire rope and hand winch. A stream of water is pumped through the rod and escapes through openings in the chopping bit. Casing is used, the upper end of which is provided with a *T* for the discharge of the cuttings and water. The drill is given a short stroke, 5 to 8 in., and is rotated by hand. A low derrick is necessary. Jetting drills used for iron-ore prospecting upon the Mesabi gave drilling rates of 25 to 40 ft. per shift for the first 100 ft. of hole and some 20 ft. per shift for greater depths. Light portable churn-drill rigs may be used with the jetting drill.

6. The Diamond Drill.—Though rotary boring tools are very old, it was only in 1864 that a French engineer, Rudolph Leshot, used diamonds, set in an annular steel bit, for penetrating rock. Rotary tools with the bits set with this hardest known abrasive substance were used then for drilling blastholes in the Mont Cenis tunnel. The general principles of the diamond drill have remained much the same. Nowadays its application is in mineral exploration, not in drilling blastholes, where it was soon superseded by the percussion rock drill.

The diamond drill consists of a rotating mechanism carrying a hollow feed screw, at one end of which is a chuck for holding the drill rod tight. The feed screw is rotated by a bevel gear containing an inner quill that slides in a longitudinal slot in the feed screw, the bevel gear meshing with its complement, which is keyed to the engine shaft. The driving power may be a compressed-air engine, steam engine, electric motor, or gasoline or diesel engine. A clutch is interposed between engine and drive shaft. A differential feed, operated by gears and by a shaft parallel to the feed screw, allows adjustments for variation in the forward advance of the drill rod which can be thrown quickly into a neutral position. The assembly of the parts, known as the swivel head, can be swung through a 180-deg. angle, thus clearing the line of the drill hole. Larger drills are equipped with hydraulic feed cylinders and are provided with a frame permitting the drill head to be racked back from the hole. In drills for surface operation both engine and swivel head are mounted upon a steel frame, which also includes a small hoisting drum for pulling drill rods. The hollow drill rod, through which passes a stream of water, is rotated at a comparatively high rate of speed. It is fed down until the bit cuts rock, when the feeding rate is regulated to accord with operating conditions. Drilling is by successive stages, determined by the necessity of removing the core. Sections of drill rod are added as the hole deepens. Figure 19 is a view of a prospecting drill.

For a long time few improvements were made in the diamond bit, which consisted of a soft steel blank in which six to eight black diamonds were set by hand. Scarcity finally so increased the price of black diamonds as to seriously restrict diamond drilling. In 1927 D. G. Longtin tried out bort or small industrial diamonds in place of the customary black diamonds. He hand-set 40 to 72 stones in each bit and reduced his costs and increased his rates of drilling remarkably. However, the hand setting of many stones in a bit proved a difficulty. In the same year J. K. Smit tried to introduce a cast bit in which bort was used, but failed. F. Koebel and J. J. Koen were more successful with a cast insert bit in 1937. F. W. Rolland took out patents in 1938 for the use of powdered metals in forming the diamond-holding portion of the bit. He used a cadmium, tin, and copper mixture fused into an alloy by heat and pressure. Thus appeared the mechanically set bit, which greatly lowered the cost. Next appeared bits consisting of a matrix of tungsten carbide and cobalt impregnated throughout the crown with 8- to 48-mesh size dia-

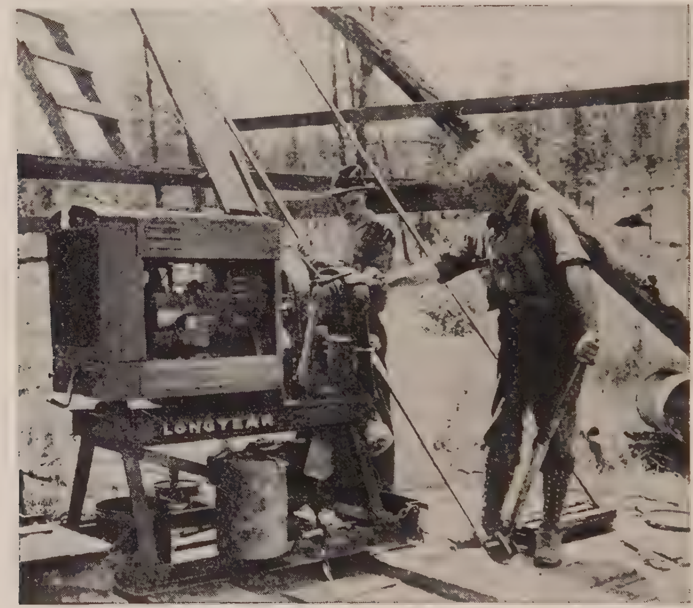


FIG. 19.—Gasoline-powered light prospecting drill on an inclined hole. (*E. J. Longyear Co.*)

mond chips. Other matrices were also used. Cast, sintered, and impregnated bits are now available in various designs, some of which are shown in Fig. 20.

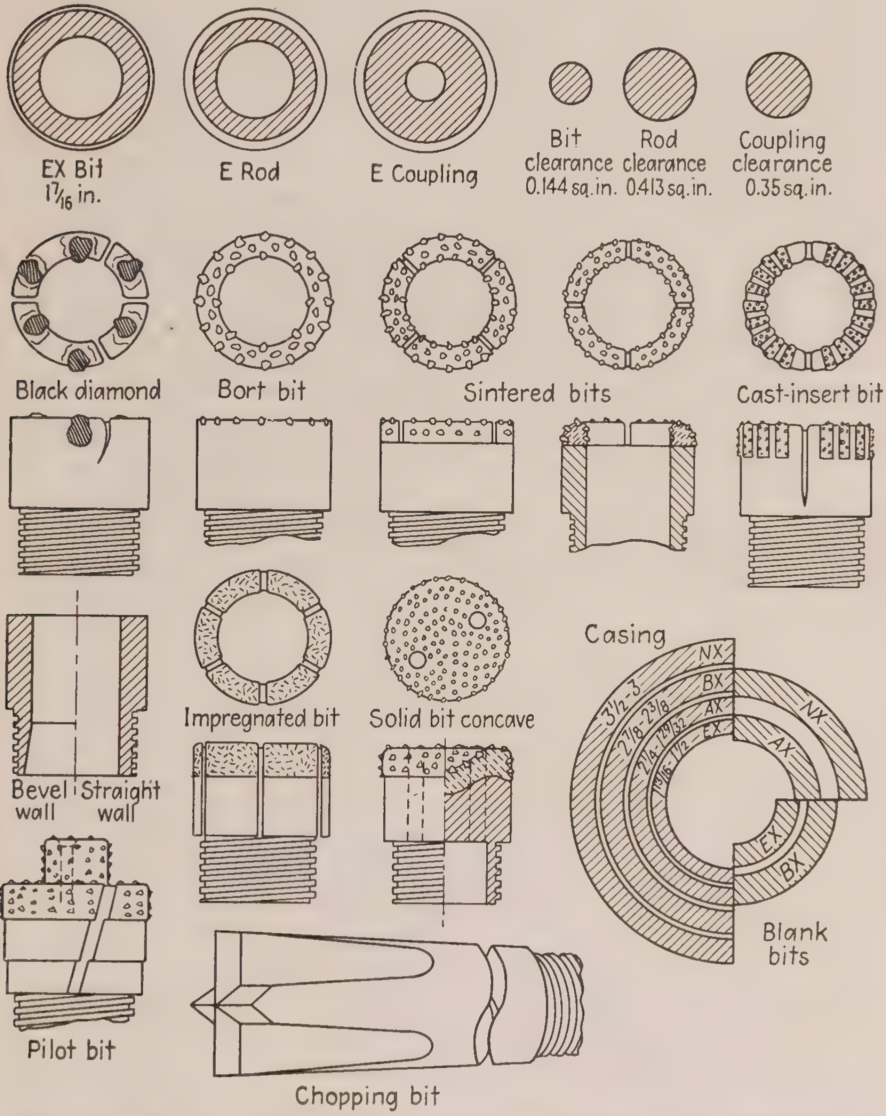


FIG. 20.—Clearances of bits, rods, couplings; some diamond-bit designs; sizes of bit blanks and casings; chopping bit.

Faster cutting speeds and lower costs for diamonds have wrought a revolution in diamond drilling. In 1930 W. J. Mitchell introduced a drill weighing only 160 lb., mounted upon a standard rock-drill column and clamp. Mitchell also experimented with coreless bits and tried out convex bits set with bort. In that year he used these coreless bits successfully for drilling grout holes at Boulder Dam. They indicated the possibility of using diamond-drill holes for blasting purposes. Early in 1929, W. I. Nelson described pillar mining at the Engels mine (California). Down holes of 20 to 207 ft. and up holes of 20 to 183 ft. were drilled in the pillars by means of diamond drills, then loaded with gelatine dynamite and Cordeau detonating fuse and successfully blasted. The technique and advantages of long-hole blasting were demonstrated.

The lightweight drill and cast, sintered, and impregnated bits have widened the field of the diamond drill. Many mines are now equipped with light drills and use them not only for exploration work but also for other purposes. In some mines, as at Noranda and Mt. Isa, light drills are used for blasthole drilling. Their practice follows that originated by Nelson. There is a strong probability that the diamond drill will find greater and greater use where long blastholes can be used to advantage, as in pillar recovery and some forms of shrinkage and sublevel stoping.

TABLE 8.—STANDARD BIT AND CASING SIZES, INCHES
(After Longyear)

Designation	O.D.	Core diameter	Bit blank o.d.	Casing		Casing bit o.d.
				O.D.	I.D.	
EX	1½	⅞	1 7/16	1 13/16	1⅝	1 27/32
AX	1⅞	1⅛	1 27/32	2¼	2	2 5/16
BX	2⅜	1⅝	2 5/16	2⅞	2 15/32	2 15/16
NX	3	2⅛	2 15/16	3½	3 1/16	3 9/16

TABLE 9.—CORE-BARREL, DRILL-ROD, AND COUPLING DIMENSIONS
(Longyear)

	EX		AX		BX		NX	
	O.D.	I.D.	O.D.	I.D.	O.D.	I.D.	O.D.	I.D.
Single-tube core barrel	1 7/16	1	1 13/16	1¼	2 9/32	1 11/16	2 29/32	2¼
Couplings—core barrel	1 7/16	7/16	1 3/16	9/16	2 9/32	⅝	2 29/32	1
Drill rods	1 5/16	27/32	1⅝	1 13/16	1 29/32	1 13/16	2⅜	2
Couplings	1 11/32	13/32	1 43/64	17/32	1 15/16	⅝	2 13/32	15/16
Number of stones to bit	185		200		235		300	

Certain size standards for bits, casing, and drill-rod threads were established in 1930 by the Diamond Drill Manufacturers' Association. Four sizes for bits, drill rods, and casing are standard, although other larger and smaller sizes are in use. Table 8 gives sizes of bits and casing. Table 9 gives dimensions of core barrels, couplings, and drill rods.

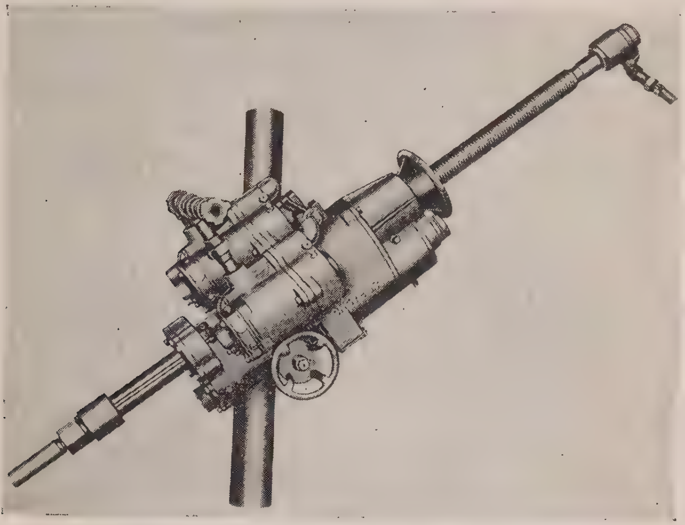


FIG. 21.—Light core and blasthole drill. (*E. J. Longyear Co.*)

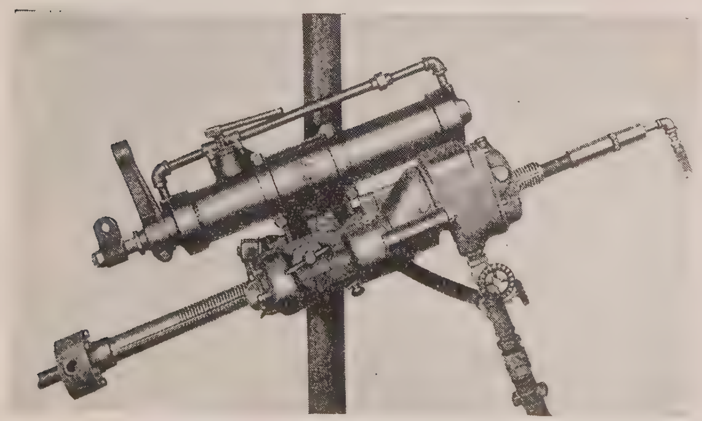


FIG. 22.—Light diamond drill with rod puller. (*Chicago-Pneumatic Tool Co.*)

The EX bit is used mainly for shallow holes up to 1,500 ft. or more. For holes 1,000 ft. or more in depth, larger bit sizes and core diameters are considered necessary. Holes of 1,000 ft. are comparatively common in exploration work, but deeper holes are much fewer in number. A drilling campaign may require many holes 400 to 500 ft. in depth, a few of 500 to 1,000 ft., and only one or two over 1,000 ft. Only recently a 3,000-ft. hole was drilled horizontally in a Grass Valley mine for exploration. Very deep diamond-drill bores are of 5,000 to 7,000 ft. and are vertical.

Larger bit diameters than the standards given in Table 8 have been employed for special purposes. At one time diamond drilling was tried out for petroleum wells in hard formations, but high cost and the development of rotary rock-cutting bits pushed the diamond drill out of this field. The largest diamond bit, 24 in. in diameter, was used in 1942 by the Mitchell Diamond Drill Company on dry-dock work.

Dimensions of light drills upon the market are given in Table 10. Figures 21 and 22 illustrate different designs.

TABLE 10.—LIGHTWEIGHT DIAMOND DRILLS, COLUMN MOUNTED

Make	Hp.	Weight, lb.	Length feed screw, in.	Rod puller		Bit speed, r.p.m.
				Stroke, in.	Weight, lb.	
Longyear:						
Gopher	9[a]	180	24	30	80	0–3,000
Prospector	3.5	248[b]	12	33	73	520–800
Sullivan:						
H.S.-15	10[c]	155	27	37	114	0–3,300
No. 6	[c]	207[b]	18	37	114	0–1,500
Chicago Pneumatic:						
No. 5	[a]	160	24	21	90	0–1,500
Holman[d]	[a]	230	24	..	...	high

[a] Signifies a compressed-air engine of the vane type.

[b] Drills marked [b] consist of separable swivel head and air motor; all others have drill and air motor integral.

[c] Refers to the Sullivan compressed-air motor, which is not of the vane type.

[d] All drills except the Holman have gear-driven variable feeds and may be swung out of line for the withdrawal of drill rods. Holman has a hand-screw feed and a pressure indicator.

A heavier drill for surface prospecting is the Longyear UG Straightline. This is a gasoline-powered drill rated at 1,750 ft. Its specifications are as follows: EX bit; weight of drill complete, screw feed, 2,000 lb.; four-cylinder gasoline engine rated 25 hp. at 1,400 r.p.m.; spindle speed with medium-bevel gears 190 to 1,350 r.p.m.; hoist 9.5-in. diameter drum, face width 6 in., and rope capacity 150 ft.; cable 0.5 in. Many other types and sizes are available.

With the diamond drill, cutting is the result of rapid rotation, pressure caused by either the feed screw or the weight of the drill rods, or both (in hydraulic-feed type drills by hydraulic pressure), and the abrasive quality and number of projecting diamond points presented by the bit. The nature of the rock mass penetrated by the drill is necessarily an important factor in determining drilling rates. Speed of rotation has been increased from 600 r.p.m. to 2,000 and even 3,000 r.p.m. Optimum speed is determined by conditions. Manufacturers are conservative in their advice about speed. At high speeds vibration increases greatly, with resultant shock and wear to the whole drill mechanism. Excessive vibration is difficult to control. Light drills are given maximum speeds of from 1,000 to 5,000 r.p.m. or more; the maximum for the heavier drills is about 2,000 r.p.m. Deep-hole drilling is restricted to about 1,000 r.p.m. There seems to be an optimum speed as well as an optimum bit pressure, but these lack accurate definition.

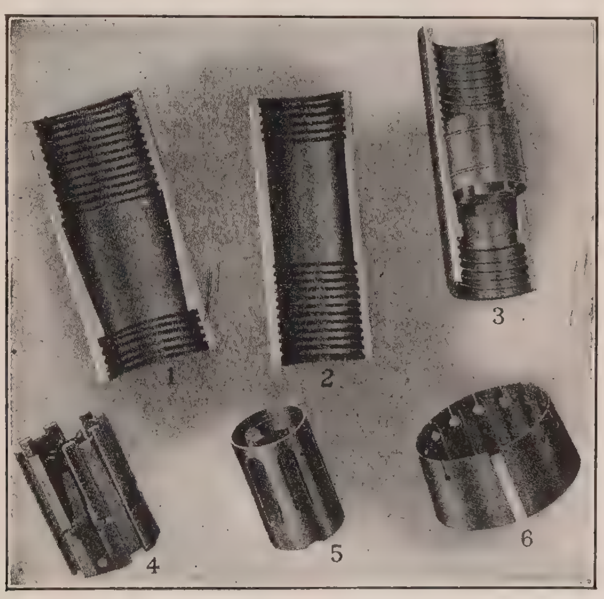


FIG. 23.—Core lifters; (1), (2), (3) portion of core barrel; (4), (5), (6) core lifter.

The coring string consists of the diamond bit, reaming shell, core lifter, core barrel, ferrule, and a drill rod made up of 5- or 10-ft. sections connected by couplings. Figure 23 illustrates the parts of the core lifter; Fig. 24, the water swivels; and Fig. 25, the drill-rod clamps. A guide rod 10 ft. in length is used sometimes immediately above the core barrel to prevent deflection. It is an oversized rod; in the E size, $1\frac{7}{16}$ in. o.d. and 1 in. i.d. The drill rod passes through the chuck and the hollow feed screw of the drill and carries a ball-bearing water swivel, to which the drill water supply is attached. Diamond bits are of two general classes, bevel wall and straight wall; the former is used with and the latter without a core lifter. The reaming shell, into which the bit screws, is set with diamonds, which maintain the gage of the hole. The core lifter is a split cylindrical ring, the section of which is wedge-shaped.

When the rods are removed, the wedge ring, or core lifter, crowds against the core, breaking it off and retaining it in the core barrel. The core barrel is 5 or 10 ft. in length, though longer core barrels have been used. The ferrule is a hollow cylindrical piece in which longitudinal water grooves permit the water to pass; its function is to maintain alignment at the rear end of the core barrel, which is supported at its front end by the bit and reaming shell. The hollow sectional drill rod transmits rotation and carries water to the bit; clearance of the E rod in a 1.5-in. hole is $\frac{3}{32}$ in. all around, which is space enough for passage of water and cuttings.

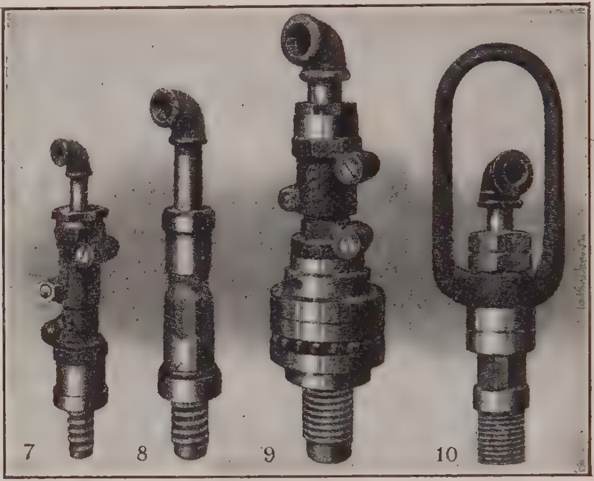


Fig. 24.—Water swivels; (7) common water swivel with stem lubricators, (8) common water swivel, (9) ball-bearing water swivel, (10) combined water swivel and hoisting plug. (*Sullivan Machinery Co.*)

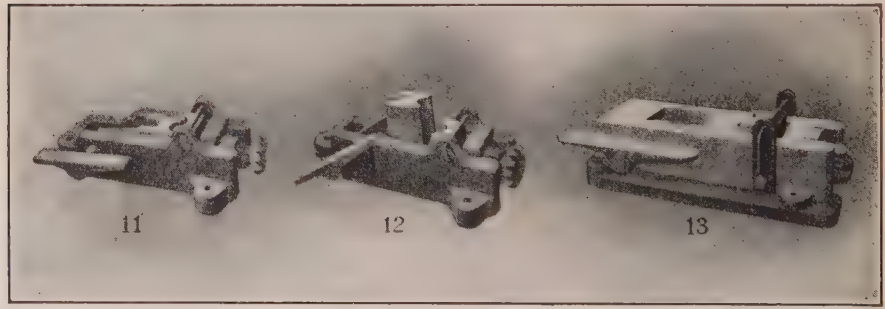


Fig. 25.—Drill-rod clamps; (11) clamp closed, (12) clamp closed, (13) heavy clamp. (*Sullivan Machinery Co.*)

The majority of bits are processed, either cast or sintered set. They are made as flat-faced, full-contour, half-contour, and thin-faced bits. They are now commercially manufactured by methods that ensure uniformity of gages. The use of low-cost bort has resulted in greatly reduced diamond cost, faster cutting, and smoother coring. Coreless or bullnose bits, used principally for grout and blasting holes, have appeared in considerable variety. Four types may be noted, the convex, the pilot, concave, and the flat-faced. W. J. Mitchell experimented with a convex bit about 1930 since which time others have experimented with different kinds. The concave bit appears to find considerable favor.

Core barrels are of three types, the single tube, the rigid double tube, and the swivel double tube, with or without ball bearing. In the single-tube type, the water passes over the core and discharges through the waterways of the bit. Thus the core is under pressure and washing effect. In the rigid double-tube type, water passes down between the outer wall of the inner tube and the inner wall of the outer tube and is discharged through small holes in the reaming shell in proximity to the bit, part of the water being discharged at the bit and part returning around the core to escape holes near the top of the core barrel. The core is protected from any washing effect, and core recovery from soft friable rocks is increased. The rigid double-tube core barrel is used without a core lifter, the driller depending upon favorable blocking near the end of a run to break the core. The swiveled double-tube core barrel permits rotation of the inner core barrel, thus causing less interference with the core and increasing core recovery. Double-tube core barrels are used where a high percentage of core is required in soft rocks. A short, rigid tube is used for coreless and blasthole drilling.

Numerous accessories are required for drill operation. A low-type derrick is used for surface work. A hoist, rope, and sheave facilitate handling drill rods into and out of holes. In deep-hole work a higher derrick reduces the number of makes and breaks, an important timesaver. Compressed-air-operated rod pullers take the place of the small hoist in underground work, although hoists are frequently used for down-hole work. The rod puller is conveniently attached to the drill support and can be used to operate in either direction by reversing the pulling dog in the offset arm. All deep-hole surface rigs should be equipped with safety clamps, which are placed at the collar of the hole and hold the drill rods during replacements of rod sections or when rechucking. For shallow down holes, lowering irons or pulling dogs are used to control the rods. At all times the rods must be held by the drill chuck, clamp, or safety dogs; otherwise they fall into the hole and a fishing job results. In the case of up holes, pulling dogs attached to a chain and near-by eyebolt are used to give control of the rods; rod pullers are also used to hold and control the movement of drill rods. Fishing, pipe, and casing tools are essential to the driller's kit.

Casing is required where broken, fissured, and ravelly ground is encountered. The casing bit is used to ream the hole to the size necessary to take the casing. Instead of casing, cement grout is sometimes used successfully to patch hole walls. Several lines of casing may be necessary in deep-hole work, sometimes as many as four. The casing has a flush joint and is made especially for diamond-drill bores. Various sizes have already been given in Table 8.

Drilling rates involve many variables, including rod speed, pressure, bit construction and size, direction of hole, formation, coring, and drill operator's experience. The mechanical condition and power of the drill play a part. Other things being equal, formation becomes the most important controlling factor. Hard, tough rocks drill slowest but yield good cores; soft rocks drill much faster but require slower rod speeds and more cautious drilling where the core is to be recovered. Jasper, tough hematite, quartz, quartzite, silicified limestone chert, diorite, granite, and the harder igneous and metamorphics give low rates; shale, slate, limestone, and sedimentaries give higher rates. Unconsolidated formations, sands, gravels, and the like cannot be handled by the diamond drill. Where such rocks are at the surface, they are drilled by a churn or jetting drill, after which the holes are cased and the diamond drill is started upon solid rock. Fissured rocks, vugs, cavities, fracture zones, and selvages delay drilling, and when casing or cementation is necessary the time required for these operations slows down drilling. Very deep holes require a much greater proportion of time for running in and hoisting drill rods. Longyear estimates 1 hr. for hoisting and lowering rods in holes 500 to 750 ft. deep; for holes of a mile in depth, 8 to 10 hr. For rocks above a hardness of 3.4 to 4 (Mohs' scale), EX or AX bits are used, but softer rocks require a larger bit size where core recovery is important. The larger the diameter of the hole, the greater the total cost. Very large diameters of core bits have been used in oil-well operations, but costs are very high.

Burned bits result from high speeds, excessive pressures, and poor water circulation. Under such circumstances sufficient heat is generated to cause the metal of the bit to flow, resulting in distortion, displacement of the diamonds, and a ruined bit. On the other hand, Coen states that at low feed rates, say 600 r.p.m. per inch of feed, diamonds sometimes polish and fail to cut. The bit must be forced up to its work at rates of 100 to 400 r.p.m. per inch of feed, depending upon formation conditions. The life of a bit may reach extreme limits of 800 to 900 ft. or it may be only 50 ft. Longtin gives the life of a bit in hard andesite as 28 to 50 ft. and in sandstone 225 to 275 ft., the diamond loss in the sandstone amounting to 2 carats per bit. Weslow gives examples of diamond-impregnated carboloy bits used on hard iron ore, including two EX bits, one of which had a life of 189.88 ft., with loss of outside diameter of 0.048 in., the other 81.1 ft., with loss of 0.032 in. Maintenance of gage determines the life of a bit. The efficiency of the reaming shell in maintaining hole gage is also a factor. Too long a use of a bit may require reaming to restore the hole diameter.

Water circulation is necessary. Pumps provide pressures from 100 to 250 lb., which suffices for most drilling. Water absorbs the friction heat and removes the cuttings. It also acts as a lubricant, although dope or grease is also used on the rods for the same purpose. Longyear estimates the amount of circulating water as 150 to 175 gal. per min. where none of it is lost in the drill hole; much larger amounts are required where open formation causes water loss.

Where core is being lost and sampling is necessary, the sludge is saved and used as a sample. The relative proportions of core and sludge, assuming 100 per cent core recovery, are 33 and 67 per cent, respectively, for the EX bit; for the AX, 40 and 60 per cent; for the BX, 50 and 50 per cent. Core loss is represented by increased sludge. Various arrangements of sludge boxes are used for catching the sludge flow as it leaves the T at the collar of the hole. The sludge for a given run is taken as a single lot, the circulating water during the run being kept low. On completion of the run, the hole is flushed clear of cuttings and the rods are removed. In case of blocking, the rods are removed and returned, and any remaining sludge in the hole is washed to the sludge box or barrel. No dope or grease is used on drill rods when sludge sampling is in progress. Surplus water is decanted from containers; the sample is then collected and placed in drying pans. Sludge samples are not accurate where the hole loses water because of fissured ground. Under such circumstances the hole is cased or cemented before sludge samples can be taken. Hole tightness can be tested for by the use of a "hole packer" and pressure water. The requirement of careful coring and sludge sampling necessarily slows down drilling rates; both are factors to be taken into consideration in estimating drilling costs and time required for a drilling campaign.

Many specific examples of drilling rates can be found, ranging from simple bit tests to average drilling rates of a campaign involving many holes. An underground drilling rate of 20 to 30 ft. per shift is often considered adequate. Hard compact andesite has been drilled at the rate of 60 ft. per shift by a coreless bit. J. Q. St. Clair gives an interesting account of a drilling campaign undertaken by a South African gold mine. Three light drills were used, a Sullivan No 6, a Mitchell, and a Boyles Bros. Bits were set with Congo-type bort—EX size with from 46 to 102 stones; reaming shells 34 to 46 stones. There were 245 holes drilled to an average depth of 76 ft., shortest 1.5 and longest 534 ft. Fifty-four per cent of the holes did not exceed 60 ft. in depth. Underground drilling totaled 17,724 ft., surface 913 ft., reaming for casing 300 ft., drilling cemented holes 166 ft.; total 19,103 ft. Time distribution, by percentages, was as follows: Traveling to work, 8.9; moving machinery, 18.5; drilling, 51.1; and miscellaneous, 21.5 per cent. Drilling rate on actual drilling time was 3.4 ft. per hr.; on total time 1.8 ft. per hr. on holes 90 to 150 ft. in depth. Forty-six per cent of the holes were up, 40 per cent down, and 14 per cent horizontal. Average core recovery was 92 per cent. Total hours was 14,152; total footage 19,103; average rate per hour 1.35 ft., or 10.8 ft. per 8-hr. shift. Total drilling time was 7,233 hr., an average at 2.64 ft. per hr. and 21.12 ft. per shift. Types of rock drilled, by percentages, were quartzite 57, shale 21, intrusive 14, and undifferentiated 8.

Drilling Salines.—Many unusual situations develop in core drilling, and both engineers and operators frequently devise expedients to overcome them. Coring in soft claylike minerals is very difficult, as water under pressure tends to cut out and erode the core. Sometimes compressed air is used instead of water, and with slow-speed drilling successful coring has been accomplished by this method. Drilling of salt beds in western Texas and New Mexico was described by J. S. Wroth in 1929. Diamond drills were used for coring the significant beds. Churn drills were used to the salt formation; after casing, core drills were used. Some modifications were made in the bit and reaming shell by which the water from a ball-bearing double-tube core barrel was brought down through the reaming shell and through holes to the cutting track of the bit without coming in contact with the core. The core was 2.5 in. in diameter. To avoid dissolving any part of the core or formation a saturated salt solution in which was dissolved 3 lb. of magnesium chloride per gallon of water was used as a circulating fluid. Core barrels held 16 ft. of core and as much as 15 ft. were pulled per round of drilling. The core lifter sometimes left as much as 3 ft. of core in the hole, but this was recovered during the next drilling round. Churn-drilling costs were $5.56 per foot. The total depth of 12 holes was 20,443.75 ft., of which 10,876.5 ft. were core drilled. Recovery was 10,737 ft. of core, or 97.91 per cent. Cost of core drilling was $12.27 per ft. Total cost of all drilling, both churn and core, was $11.53 per ft. The drilling was done by the U.S. Bureau of Mines under contract.

Cementing Walls of Diamond-drill Holes.—It has been the practice to cement broken formations ever since the early days of the diamond drill. In 1909 J. M. Fox described cementing practice at Tonopah, Nev. He found quick-setting cements better than ordinary kinds. In comparatively shallow holes, cement grout was poured in as thick a mixture as possible. For deeper holes, thick cement grout was poured into paper tubes 12 to 14 in. in length. The tubes were dropped into the hole, the hole having been blown dry wherever possible before cementing. The cartridges were tamped by the drill rod. Other operators have introduced cement grout through the drill rods, using a pressure vessel to force the grout through. Rods are washed out after use in this manner. J. S. Wroth used cement for sealing porous salt formations by lowering thick cement grout into the well in special bailers. Where cavities existed, dry cement was used in cartridges of heavy paper, 2 to 3 ft. in length, and lowered to the bottom of the bore and tamped with the drill rods. Cement cartridges were thus used in both dry and wet holes. In the latter case the paper of the cartridge was shellacked to prevent disintegration on the way to the bottom.

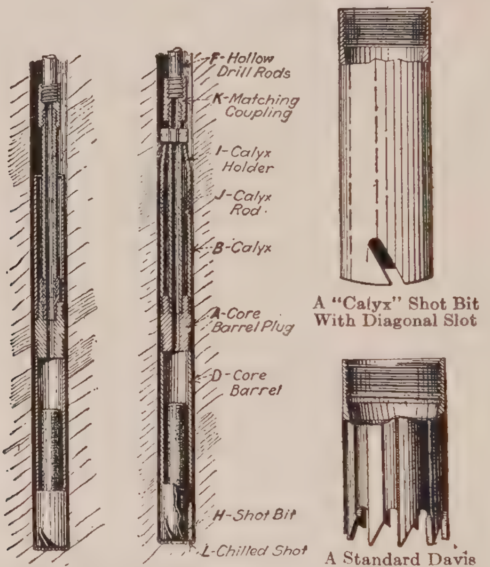
Wear of Diamonds.—Stones in black-diamond bits are from 1 to 4 carats in size, and proportionally greater wear may be expected from them than from the smaller bort stones, which run from 16 to 20 stones per carat and upwards. L. T. Wright reported black-diamond losses as high as 0.08 carats per ft., in hard felsite 0.04 carats per ft. and in hard silicified eruptives 0.01 to 0.025 carats per ft. On the Rand, South Africa, diamond wear was 0.02 carats per ft. in hard quartzite; on the Michigan iron ranges wear was 0.012 to 0.027 carats per ft.; in the Boundary district, British Columbia, 0.006 to 0.03 carats per ft. in hard eruptives. Bort bits with many more cutting edges probably wear twice as fast as the black diamond, but they cut faster and cost less.

Coreless bits cut the full diameter of the hole, the standard EX bit about 66 per cent, and the thin-faced EX about 60 per cent. The respective wear may be expected to be in the same proportion. A. V. Corlett and G. D. McLeod gave examples of hand-set coreless bits with 325 to 350 stones per bit (60 stones per carat and 5.5 to 6.5 carats per bit) that drilled 85 to 90 ft. and recovered about 40 per cent of the stones; some bits drilled as high as 300 ft. after being patched. W. R. Dunbar described Hollinger practice in which square-faced EP and ES bits, cores ⅝ and ⅞ in., gave respective runs of 88 to 145 ft. and 111 to 369.5 ft. The smaller bit was set with 2.07 carats and the larger with 3.19 carats. Stones ran 70 to the carat.

Diamond-drilling Costs.—Diamond-drilling costs vary in accordance with wages, diamond costs, and local conditions. Specific examples of costs are to be taken with caution. Contracts are made by some Nevada mines at $1.75 per foot for 1,000 ft. or more, the mine supplying compressed air and the contractor his own equipment. L. A. Norman breaks down drilling costs at the Old Eureka mine (California) as 34 cents per foot for core-bit diamonds, reaming shells, and casing bits, and $1.13 per foot for labor, but he does not give power or other costs.. Each shift averaged 20 ft. At another mine four shallow holes in quartzite aggregating 158.8 ft. gave a total cost of about 80 cents per foot; 60 per cent of the time was used for drilling and 40 per cent for moving and miscellaneous. Cost percentages were apportioned as follows: drill runner

38, helper 4, carbon loss 16, bit setting 32, power 9, and lubricants and tools 1 per cent. Before the war diamond-drilling costs were about one-half of those prevailing in the "black diamond" period. Very low costs were obtained in blasthole drilling, on the order of 50 to 100 cents per foot. Postwar costs are uncertain but may well be double the costs published in technical articles between 1930 and 1943.

7. Chilled-shot Drilling.—The rotating and drill-rod arrangements are similar to those used in diamond drilling. Larger holes are drilled and bigger cores obtained. The bit is soft steel and is attached to the core barrel, which is in turn attached to the hollow drill rods. Above the core barrel is an annular open shell, which receives the coarsest cuttings. Figure 26 illustrates the cutting parts. Core lifters are not used; the core is detached by introducing a quantity of small gravel into the feed water. This wedges between the core and bit, and the rotation of the bit breaks the core. By the use of a drag bit the formation of a core can be avoided. Chilled shot is used as an abrasive and is introduced at regular intervals through a tee swivel, one branch of which serves for the introduction of the shot and the other for the water. The quantity of water must be carefully regulated, since too much water will wash the steel shot out from the bottom of the hole. The arrangement for introducing the shot is shown in Fig. 27.



A "Calyx" Shot Bit With Diagonal Slot

Standard "Calyx" Tools
Without Holder 3½" Bore, 2½" Core
With Holder For Cores Greater Than 3.5"

A Standard Davis Cutter Used on the "Calyx" Drill

Fig. 26.—Cutting bits used in chilled-shot drilling. (*Ingersoll-Rand Co.*)

The first cost of a chilled-shot drilling rig is comparable with that of a diamond drill. Commercial sizes range from 2⅝-in. bore and 1⅝-in. core for the smallest to 20.5-in. bore and 18⅝-in. core for the largest.

The rate of drilling is illustrated by the following examples:

In marble	33 ft. in 10 hr.
In sandstone	8 to 45 ft. per day
In granite	10 to 12 ft. per day
In moderately hard andesite	1 ft. per hr. up to 40 ft. per day

Where fissured rock or caving ground is encountered, drilling rates are extremely irregular. Such bores must be cased or else cemented. When they are cased a smaller drill is used for the advance.

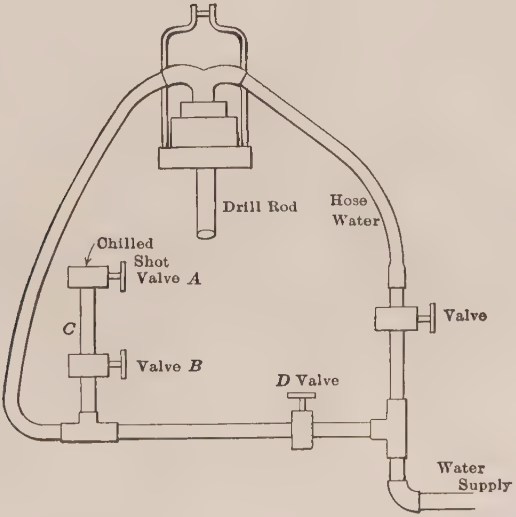


Fig. 27.—Device for introducing chilled shot into drill rod. Valve *B* is closed and chilled shot introduced through *A*. *A* is closed, *D* opened and then *B*. The water forces the shot through the swivel connection and into the hollow drill rod.

J. B. Pullen describes calyx-drill operations in pilot-hole winzing and shaft work at the United Verde mine. The holes were 48 in. in diameter, and depths up to 281.5 ft. were bored. A total of 10 holes gave a length of 1,450 ft. Holes more than 100 ft. deep cost less than the average in raises. A three-man crew was required. Weight of the bit was 4,765 lb., speed 40 r.p.m.; 2.76 ft. were drilled per shift. Drilling speed was 14.1 in. per hr.; drilling time was 20.4 per cent of total time; 22.8 lb. per ft. of No. 8 chilled shot was consumed. Bit wear was 6.6 in. per 100 ft. of hole and cost was $10.10 per foot. A 40-hp. motor was used for rotation and a small hoist for handling the drill and core. The headroom required above the drill was 22 ft.

Calyx drills have also been used in the Butte district for ventilation of bottom levels and for transfers of waste rock. A 36-in. drill was used, and it is proposed to use 48-in. diameter drills in future work. Figure 28 illustrates the large calyx drill.

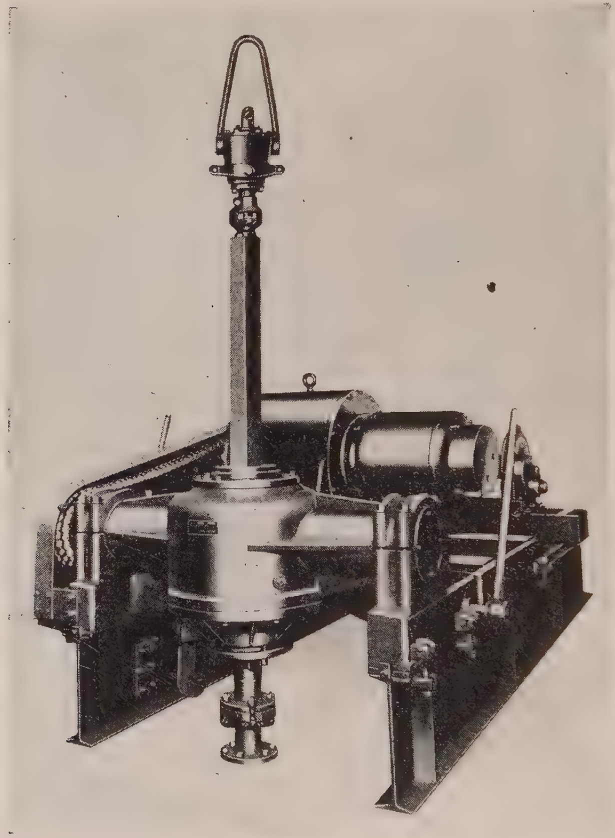


Fig. 28.—Calyx or chilled shot drill. (*Ingersoll-Rand Co.*)

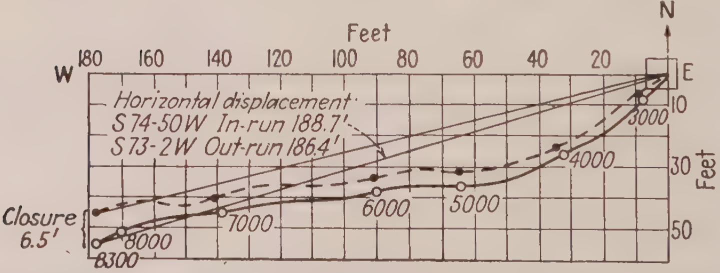
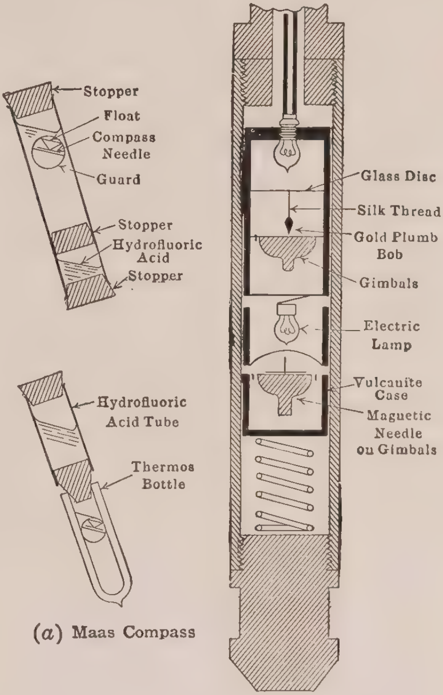


Fig. 29.—Horizontal-projection in- and out-survey of oil well, 8,300 ft. in depth. (*G. L. Kothny.*)

Deflection of Boreholes.—Horizontal and inclined diamond-drill holes over 300 ft. in depth may deflect in both horizontal and vertical planes; vertical holes may also deflect but usually not to the same extent. By surveying 255 rotary-drilled holes (oil wells) Alexander Anderson found that the average deviation from vertical for 500-ft. holes is 2 deg.; for 1,000-ft., 3.08 deg.; for 3,000-ft., 6.7 deg.; for 6,000-ft., 22.4 deg. Average horizontal distances from mouth to bottom of hole are, respectively, 8.7 ft., 30.2 ft., 119.5, and 589.4 ft. Thus the drilling depth is not the vertical depth of a bore, and the position of the bottom of the hole is problematical. Figure 29 is the horizontal projection of a gyroscopic survey (Kothny) of a deep well, the individual curves representing the inrun and the outrun.

Diamond-drill holes indicate vertical angular displacements varying from a minimum (in one set of observations) of 2.5 deg. at 500 ft. to 47.7 deg. at 4,000 ft. Horizontal displacement ranged from a minimum of 10 ft. at 500 ft. to a maximum of 1,485 ft. at 4,000 ft.

Many conditions affect the course of a bit, such as the alternation of strata of varying degrees of hardness and dip, fault planes, shear zones, and pronounced contact zones. Bending of drill rods and careless operation of the drill are also factors. Holes up to 300 ft. in depth may be presumed to be on course; but the deeper the hole, the greater the need for determining its position. A large number of instruments and several techniques have been developed for surveying drill holes. All depend upon the measurement of the dip, its direction, and the bearing of the axis of the hole at a series of consecutive points along its course. By plotting the data in relation to measured depth a curve may be drawn connecting points along the approximate course of the hole in both horizontal and vertical projections.



(*a*) **Maas Compass**

(*b*) **Lower Part of Oehman Payne-Gallwey Surveying Instrument**

Fig. 30.—(*a*) Maas compass, (*b*) Oehman, Payne-Gallwey surveying instrument.

Two surveying instruments will be described. The Maas patent drill-hole compass in its simplest form consists of a 6-in. open-ended glass tube of 1⅛ in. outside diameter. The tube is divided into an upper and lower compartment, as shown in Fig. 30*a*, by a ½-in. rubber stopper. About 1 in. of dilute hydrofluoric acid (12 water to 1 acid) is placed in the lower part. A 1⅔ per cent solution of gelatine is placed in the upper part and a small compass attached below a float is dropped into the solution. A soft-rubber stopper is inserted and the glass tube is placed in a bronze case. Several lengths of brass drill rods (each 10 ft. long) are attached to the bronze case. The instrument with the drill rods is then lowered to a given depth and is allowed to remain long enough for the gelatine to solidify and for the hydrofluoric acid to etch a line. It is then removed and the position of the north and south points of the compass needle is marked on the outside of the tube. The tube thus constitutes a record of the position of the bore and the dip from the vertical. By placing the tube in a goniometer the value of the horizontal and vertical angles can be determined. It is necessary to correct the dip angle for capillarity. The time required for a 1⅔ per cent solution of gelatine to solidify is as follows:[1]

Glass tube with paper wrapper, 6 by 1⅛ in., 20 min.
Glass tube with paper wrapper, 1⁄16 in. thick, 6 by 1 in., 30 min.
Thermos bottle with wrapper, 1⅛ in. outside, 50 min.

The thermos bottle is used where a longer time is required to get the tube to its position in the bore.

The Oehman and Payne-Gallwey instrument is described by J. I. Hofman in the *Transactions* of the Institution of Mining and Metallurgy. The lower half of the instrument is illustrated in Fig. 30*b*. It consists of a plumb bob suspended over a gimbal-supported carrier on which a disk of sensitized paper records the shadow of the plumb bob suspended over it. A compass needle on a similar carrier likewise records its shadow upon a sensitized paper disk. Both instruments are in vulcanite cells. Small lamps are so placed as to flash at a predetermined interval by a clock mechanism and dry batteries, which are contained in the upper half of the instrument. A brass case surrounds the instrument and a 10-ft. brass rod separates it from the drill rods. A series of dip-and-course measurements are taken at known depths, and the approximate line of the drill hole is indicated by plotting.

In *Report of Investigations* 3773, *U.S. Bureau of Mines*, L. A. Dahner and C. J. Cohen describe a method for surveying drill holes by the use of the etch tube and the maintenance of the orientation of the drill rod as it is lowered into the hole. The axis of the etch tube is in a fixed position relative to the container and the tube is marked with a reference diameter.

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[1] Letter from E. E. White, Ishpeming, Mich.

Two alignment clamps are used, the first being placed upon the container in alignment with the tube position and the drill rod attached to the container. The second clamp is then placed upon the drill rod and aligned with the first, a reference mark being established common to the alignment sighting slots of both clamps. Next, the first clamp is removed and the container and drill rod are lowered with the alignment maintained according to the reference mark. A second section of drill rod is then attached, the first clamp is attached to it and aligned with respect to the reference mark and the second clamp, the second clamp is removed, and so on until the tube reaches the survey point, when it is allowed a sufficient time interval for the acid to etch a ring before its removal. The attitude of the ellipse etched upon the tube at each point with reference to the rod orientation at the collar of the drill hole can be observed. The clamp consists of a 20-in. length of 1¼-in. angle iron welded to a rod clamp, the arm having a sighting slot milled parallel with its edge.

Surveying Oil Wells.—Instrumental equipment for oil-well surveying stems from the Maas and Oehman instruments but other ideas have been adopted also, and a large variety of well-developed equipment is available. Service companies have been formed to cover all the details of well surveying, and such companies have devised newer and better instruments. G. L. Kothny has described instruments brought out by the Sperry-Sun Well Surveying Company of Philadelphia. The following summary is from his paper on the subject: One of the most widely used is the syphonic inclinometer, Fig. 31, a single-shot instrument that is lowered to a given position in the bore and allowed to stand a short interval, during which a marking fluid records its level upon a cylindrical chart within its recording chamber. The instrument is withdrawn, the chart is removed, and its reading is reduced to degrees.

The EC inclinometer, Fig. 31, consists of a plumb-bob pendulum provided with a floating platinum needle, which is continuously in contact with the upper surface of the record disk, the pendulum being free to swing in any direction. Dry batteries supply a continuous flow of electricity through the pendulum to the record disk when the instrument is placed in its protective casing. When the instrument is lowered to a given position and held for at least a minute, a record is obtained on the disk by electrolytic action. This record is a white dot, the diameter of which is determined by the length of time the inclinometer is held at any one point in the bore. The range of the inclinometer is 6 deg., and by properly varying resting periods three inclination records may be obtained by one run. The outside diameter of the protective casing is 1⁵⁄₁₆ in. and its length 7 ft.

Kothny also describes a directional single-shot instrument for use in open holes, which consists of a timing clock, batteries and light bulb, a floating light-transparent compass, an inclination unit, and a photographic film for recording both the position of the compass and the cross hairs of the inclinometer. To accomplish the single record, a lens is used between the compass and the inclinometer, throwing the image of the compass upon the film surface, which is just below cross hairs of the inclinometer unit. The instrument, Fig. 31, is enclosed in a nonmagnetic casing. Only one reading can be taken at a time, but a daylight film loading device enables record films to be quickly changed.

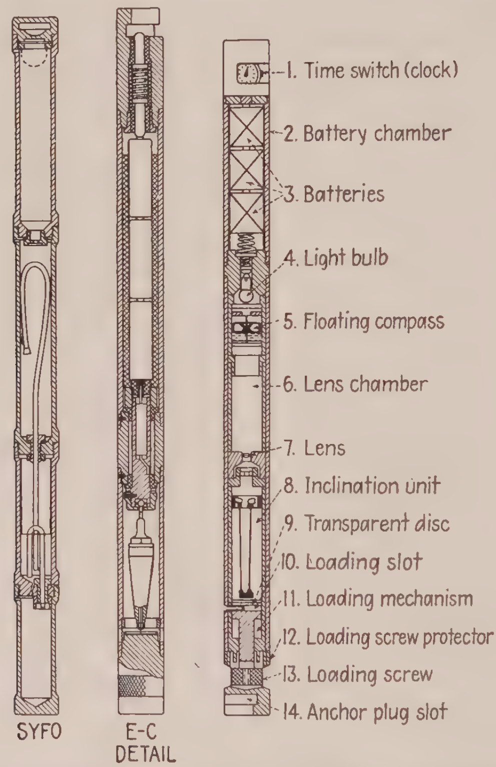

Fig. 31.—Oil-well surveying instruments—left, Syfo clinograph; middle, EC inclinometer; right, HK directional single-shot. (*Sperry-Sun Well Surveying Co.*)

A multishot instrument is also described. It consists of a gyroscopic and photographic recording, directional indicator, inclinometer of the bubble or box level type, a timing clock, a thermometer, and other accessories. A movie film enables numerous sequential records of the instruments to be taken during moving in and moving out of the bore. The recording interval may be 100 ft., but accurate surface timing is necessary for correlating depth.

Deflecting Boreholes.—Boreholes may be deflected but not without difficulty. J. I. Hofman described a method in *Transactions*, *Institution* *of Mining and Metallurgy*, vol. 21, p. 485. Figure 32 is taken from this article. The lower wedge is dropped into the bore and secured by the projecting teeth upon the bottom pipe. Its orientation is determined by a lead cup attached to drill rods and a modified Oehman instrument. The pilot wedge is oriented upon the end of the deflecting wedge and when lowered into the hole coincides in position with the lower wedge. The bit is guided upon its new course by the curved surface of the deflecting wedge. In petroleum-well work much use is made of holes that are deflected at a predetermined vertical depth. For this technique, known as whipstocking, ingenious tools based on the Hofman method have been devised.

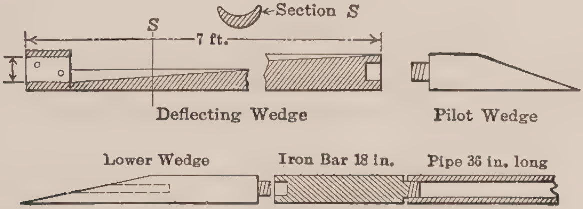

Fig. 32.—Deflecting wedge.

Miscellaneous.—Deep-hole prospecting, described by C. A. Dobbel in 1925, *Transactions, A.I.M.E.*, vol. 72, p. 677, won a place in mining operations for a considerable length of time. Long holes were drilled by means of a hammer drill, equipped with power rotation and sectionalized hollow drill steel. A swivel connection on the chuck section of the drill steel provided for water flow in place of the inner water tube. The cuttings and accompanying water flowed through a T connection and piece of casing at the mouth of the drill hole into settling tubs or barrels. A detail of the swivel and steel drill rod is shown in Fig. 42 (see Chap. IV). Holes were drilled up to a depth of 200 ft. The best drilling conditions were secured when the holes ranged from 20 to 30 deg. above the horizontal. Cuttings were dried and sampled for analysis or assay. The principal advantage of this method of boring was the utilization of the everyday mine tools in place of specialized equipment. Improvements in diamond-drilling practice and equipment were the important reasons for the displacement of deep-hole drilling by the more convenient diamond-drilling method.

Many ingenious tools have been invented for taking cores in oil-well drilling. It is possible, during the drilling of an oil well, to take formation cores at any desired point. An example is the Elliott wire-line core drill, which may be used as an accessory to the rotary drill; another is the Globe wire-line drill. Both corers require a composite cutter on the drill pipe, the principal features of which are an inner plug or center bit and an outer bit that cuts the annular groove on the outside of the core. When coring is necessary, the center bit is removed by a wire-line overshot and the coring accessory dropped through the drill pipe. Drilling is resumed until sufficient core, about 10 ft., has been obtained. Then the coring accessory and core are retrieved by the wire-line overshot, the center bit is dropped into place, and regular drilling is resumed. With these and similar devices it is possible to obtain a core without withdrawing and replacing the drill pipe. The Baker coring tool is applicable to cable or churn-drilling rigs.

Side-wall sampling of oil wells in comparatively soft rocks has been accomplished in two ways. The first is an attachment placed upon the end of the drill pipe and lowered to the sampling point. Hydraulic pressure is applied, forcing sampling or coring tubes of hardened alloy steel against the bore; the weight of the drill pipe is slowly placed upon the tool, driving the tubes into the wall; the drill pipe is lifted and removed. The description applies to the Baker rotary wall sampler. The second method makes use of a long metal cylinder in the sides of which are short cylindrical openings or gun barrels. Short hardened steel core tubes, each attached to its wire, and a powder charge are inserted in each barrel. There may be from 6 to 18 barrels in the metal cylinder, which is suspended by an electrical cable connected with firing wires at each side barrel. The cylinder is lowered to the sampling point and electrical contact ignites the respective charges, shooting the coring tubes into the side walls of the bore at spaced vertical intervals. Each tube is held by its wire and may be retrieved when the gun is removed; some are lost. The diameter of the core sample is ¾ in. and its length 1½ to 2½ in. G. L. Kothny states that from 70 to 90 per cent of the shots are successful.

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CATALOGUES:

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"Buda Earth Drill," The Buda Company.

CHAPTER IV

DRILLING FOR BLASTING PURPOSES

Hand Methods.—The hand methods are the pointed bar, auger, crowbar and posthole shovel, hammer and drill, and churn drill. The use of the pointed bar is restricted to comparatively soft materials, such as hardpan, compact earths, very soft shales, slightly cemented gravels, gouge material, and shattered vein filling. For surface work a heavy pointed steel bar 1 to 1.5 in. in diameter and from 3 to 6 ft. long is driven

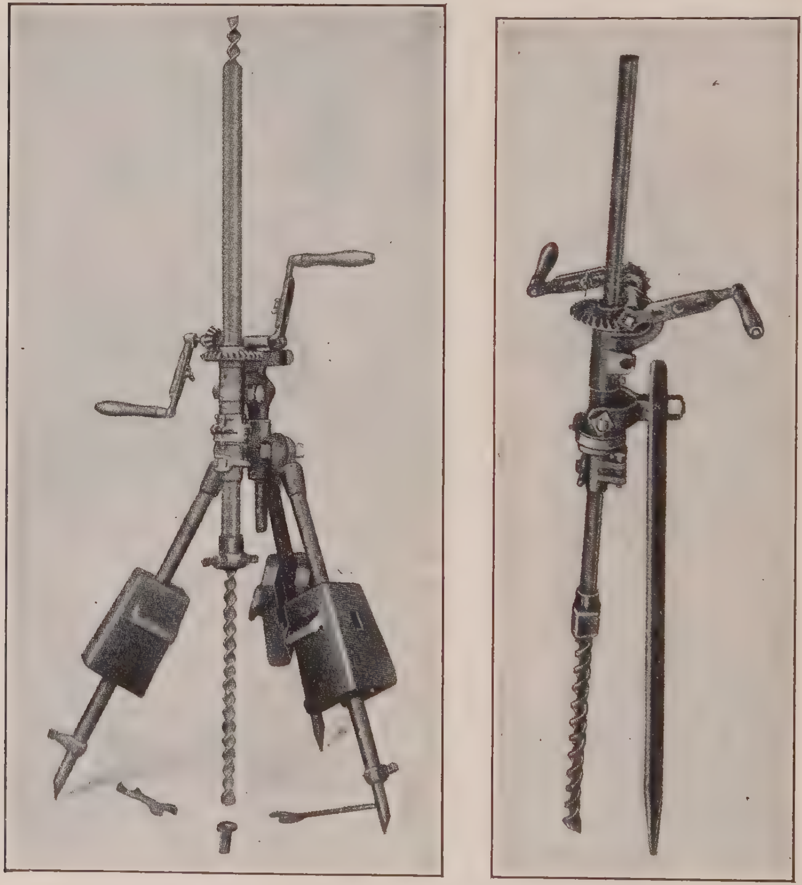

Fig. 33.—Rotary drills.

by heavy sledges. It is pulled out with a pry pole or railroad jack. For underground work a light pointed steel bar, 0.5 to 0.75 in. in diameter and 4 ft. long, is used to pick out the hole in the soft or broken material. For soft to moderately compact earthy materials the ordinary posthole type of auger is used. For harder material, such as gypsum, clay, shale, and soft slate, drills shown in Fig. 33 are most advantageously used. The tripod mounting is employed for surface work and the vertical column with screw jack for underground work. In place of the latter a horizontal bar is wedged into a small hole in the face immediately below the drill hole. For holes 10 to 15 ft. deep and 10 to 12 in. in diameter, the crowbar may be used to break up the material at the bottom of the hole and a small posthole shovel used for its removal. The method is applicable to soft rocks and compact earthy material.

For rocks of medium to extreme hardness the churn drill (jumper) and hammer-and-drill methods are applicable. The churn drill is a heavy steel bar 1 to 1.5 in. in diameter and 10 to 20 ft. long. The end is shaped into a cutting edge or bit. The drill is lifted and allowed to fall. It is rotated slightly each time. Holes may be drilled from 15 to 30 ft. in depth. The drill is principally used for "down" or vertical holes. "Uppers," provided they are not too deep and are in moderately hard rock, can also be drilled. For holes of moderate depth and diameter and any direction the hammer-and-drill method is used. It is an all-round method for small-scale mining operations. While it is possible to drill relatively deep holes by this method, it is not economical. For single-hand work the depth limit may be taken as from 3 to 4 ft., and for two- and three-handed work, 6 to 8 ft. The starting diameter of the drill hole is 1.5 to 1.75 in. and the bottom diameter 1 to 1.25 in. For single-hand drilling the use of a 4- to 4.5-lb. hammer and a 3/4-in. drill steel is common practice; for double-hand, 8- to 10-lb. hammers and 7/8- to 1-in. steel are used.

Rate of Drilling.—The rate of drilling is subject to wide variations. Gillette[1] gives as the result of his observations the rate and cost of drilling[2] 1.5-in. holes as follows:

	10 hr., ft.
Granite	7
Trap	11
Limestone	16

In crosscutting hard diorite a three-man crew drilled two 1 3/4-in. holes, aggregating 7 to 8 ft., in a shift of 8 hr. At a wage of $4 per day for each man the drilling cost equaled $1.50 per foot, not including sharpening. Some holes required 100 steels. Approximately 500 lb. of steel was required at the face. The foregoing is an extreme case that came under my observation.

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[1] GILLETTE, "Rock Excavation," p. 20.

[2] Three men in drill crew, two striking and one holding.

Power Drills.—Compressed-air and other power-operated drills have displaced manual methods except under pioneer conditions and where incidental work and difficulty of access prevent the use of such equipment. The detachable bit and the portable compressor have made possible much prospecting and exploration work formerly carried out by manual methods. The following classification summarizes various types and kinds of drilling machines used in drilling holes for blasting purposes:

1. Well drills
    - *a.* Churn drill
    - *b.* Rotary drill
2. Hammer drills, mounted—drifter type
    - *a.* Screw feed, hand operated
    - *b.* Power feed, engine operated
3. Hand-held hammer drills (plugger; sinker)
    - *a.* Heavy, for shaft work
    - *b.* Medium weight
    - *c.* Lightweight
    - *d.* Drifter-sinker type
    - *e.* Auger type
4. Stoper hammer drills
    - *a.* Hand rotated
    - *b.* Automatic rotation
    - *c.* With stoper support
5. Rotary drills
    - *a.* Hand-held
    - *b.* Column mounted
    - *c.* Car mounted
    - *d.* Automatic drill (trackless mines)
    - *e.* Horizontal strip borer
    - *f.* Diamond drill
6. Piston drill, compressed-air operated; obsolete for most mining purposes but supplied for submarine blastholes

Churn Drills, 1*a*.—Blasthole churn drills are of the self-traction type. The tread-tractor design prevails, although four-wheel steel-tired or rubber-tired trucks are also used and are hauled by truck or tractor to drilling positions. Gasoline engine, diesel, or d-c and a-c motors may be power units. The general features of the churn drill have been described in the preceding chapter. Hole diameters are from 6 to 12 in., which dimension indicates the diameter of the bit used, but blastholes are actually an inch or more greater in diameter than the bit used. Water is used in drilling and the cuttings are removed by bailing. Three sizes of churn drills are given in the accompanying table.

CHURN-DRILL SIZES
(*Bucyrus-Erie*)
(All self-traction tread)

Number	Bit diameter, in.	Weight of drill string, lb.	Weight of machine, tons	Maximum stroke, in.	Engine hp. & type
22-T	6	1,539	6.8	36	10, E
35-ft. derrick	6⅝	1,539			29, G
29-T	6	1,693	11.4	44	20, E
38-ft. derrick	6⅝	1,898			50, G
	8	2,508			45, D
	9	2,626			
	12	2,930			
42-T	9	4,300	22.6	48	40, E
47.5-ft. derrick	10	5,100			80, D
	12	5,100			

E—electric motor; G—gasoline engine; D—diesel engine.

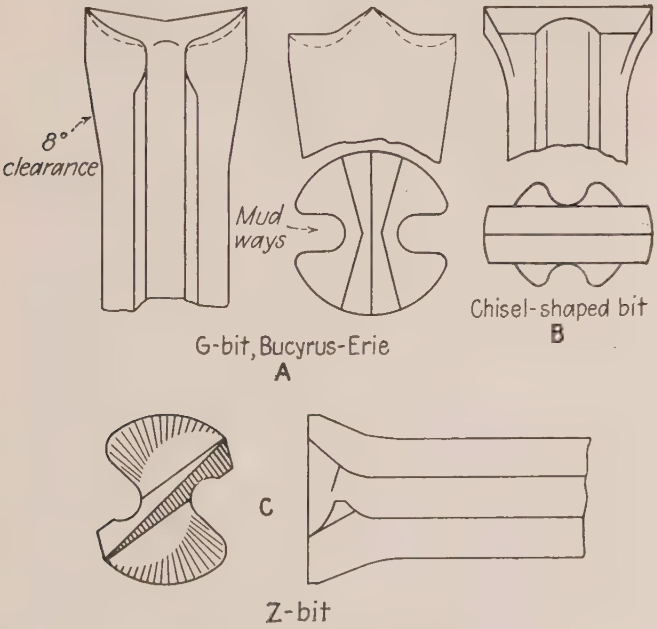

FIG. 34.—Churn-drill bit designs.

The churn drill is principally employed in open-pit and quarry work for down holes 30 to 200 ft. in depth in soft to fairly hard rocks. Figure 34 shows several designs of churn-drill bits. In very hard rocks drilling rates are often slowed down to a prohibitive extent. At Morenci (Arizona) heavy churn drills of the self-traction type drilled 85 ft. per shift and made 60 ft. per drill bit. At the United Verde churn drilling was abandoned when drilling rates dropped to 3 to 5 ft. per shift in unaltered diorite. In altered diorite, however, the drilling rate was 15 to 20 ft. per shift. At other open-pit copper mines churn drilling rates range from 50 to 90 ft. per shift for 9-in. holes. In the open-pit iron mines on the Mesabi Range churn-drilling rates are from 15 to 50 ft. in rock up to 50 to 175 ft. in ore. In limestone drilling with bits 4 in. in diameter and holes 20 to 60 ft. in depth, an electrically operated churn drill averaged 120 ft. per 10-hr. shift. At another limestone quarry electric drills averaged 3.62 ft. per hr. with $6\frac{1}{4}$-in. bits. With 9-in. bits limestone drilling ranges from 60 to 125 ft. per shift.

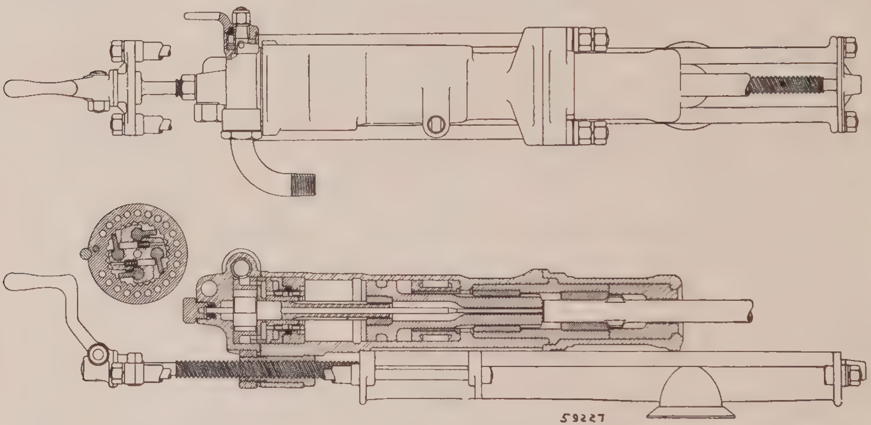

FIG. 35.—Drifter drill of screw-feed type—top view and longitudinal section. (*Ingersoll-Rand Co.*)

Rotary Drills, 1*b*.—These are not used in open-pit operations for blastholes, but a small portable type of rotary has been used for drilling seisomograph blastholes 50 to 250 ft. in depth. It is mounted on a four-wheel rubber-tired truck and is operated by the gasoline engine of the truck. Mud fluid is used and a slush pump is part of the equipment. Drill pipe is in 10-ft. sections for convenience in handling.

Hammer Drills, Drifter, 2*a*, 2*b*.—Hammer drills are compressed-air operated. The principal parts are a cylinder, piston with striking head, valve, ratchet and riffle bar, chuck, water tube and its connections, and throttle. The drill (Fig. 35) is mounted in a shell that is provided with a feed screw or other means for feeding the drill forward and maintain-

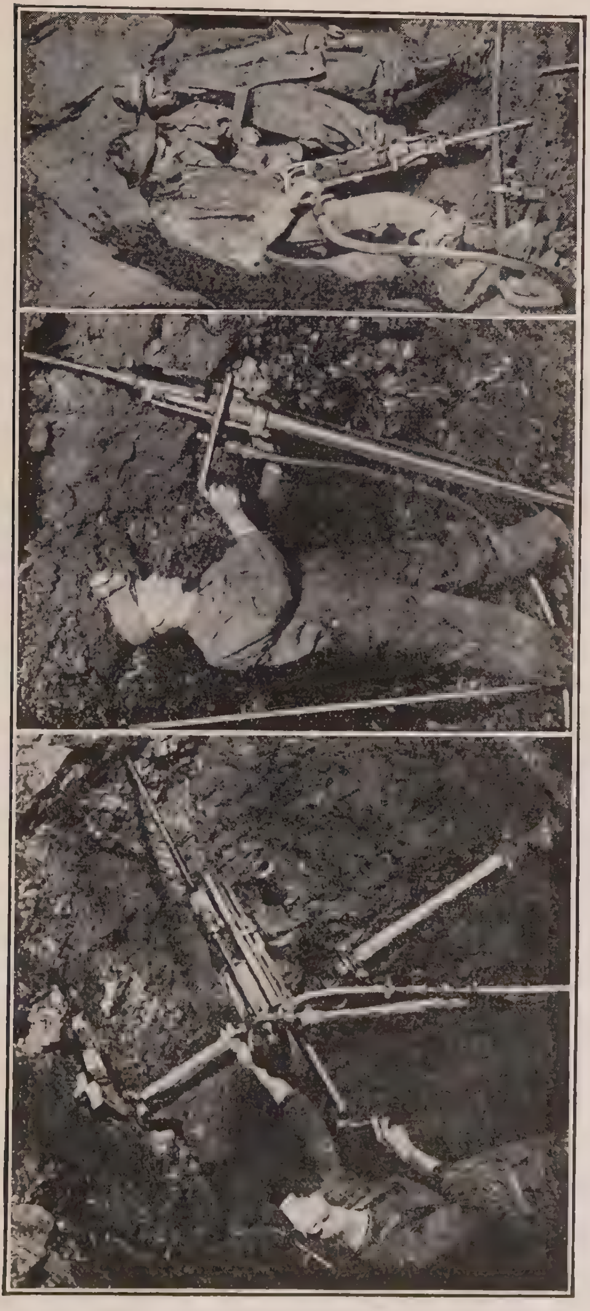

FIG. 36.—Drifter—stoper—plugger. *(Denver Rock Drill Co. and Sullivan Machinery Co.)*

ing its alignment. The shell is provided with a conical base for attachment to column, bar, tripod, or other support. Two types of shells are in use, one with a fixed cone and the other with a sliding cone. Water or water and air are carried through the hollow drill steel to the bit or cutting part. The hand-operated drill (Fig. 36) is provided with a crank for rotating the feed screw (2*a*); the auto feed is provided with an engine integral with the drill or attached to the feed screw (2*b*). All feeding arrangements provide for advancing the drill as the bit cuts and for the retraction of the drill when steel changes are necessary and all admit of adjustment. The power feeds give faster operation and quicker steel changes. The Turbro drill, which provided for power rotation of the drill steel by a separate engine instead of the riffle bar and ratchet, possessed some advantages but not enough to compensate for the greater number of parts, and is no longer manufactured. All the other types use a riffle bar and ratchet to rotate the drill steel on the return stroke of the piston. This is accomplished through the chuck that holds the end of the drill steel. Hole aligners for starting or collaring holes are attached to the front end of the feed shell and may be sprung back when not required. Drifters can be used to drill holes in all directions.

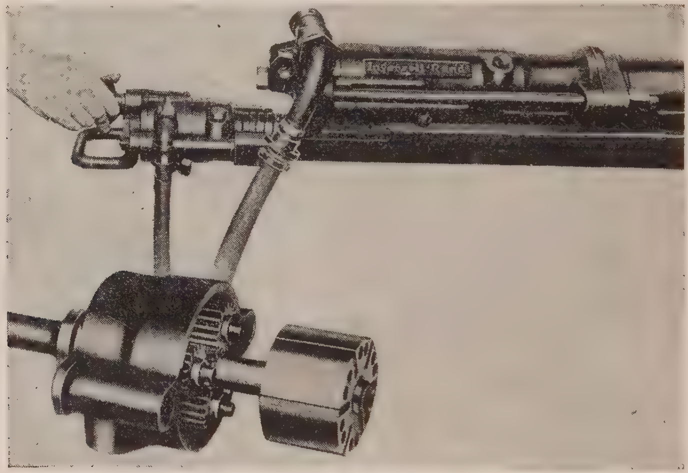

Fig. 37.—Drifter drill of power-feed type. (*Ingersoll-Rand Co.*)

Sizes and weights of drifters vary, but they may be classified as follows: Light drills, weighing from 117 to 130 lb., with 3-in. cylinder diameter; medium drills, weighing from 140 to 156 lb., with 3.5-in. cylinder diameter; and heavy drills, weighing from 190 to 298 lb., with 4-in. cylinder diameter. The usual feed length is 24 in. but in the heavy drills may be from 30 to 42 in. Power-feed drifters (Fig. 37) weigh somewhat more than the weights given above. Chuck construction varies but includes chucks for lugged steel, for hexagonal or quarter-octagon steel (a tappet being required), and for collared hexagonal or quarter-octagon steel. The principal manufacturers of drifters and other types of rock drills are Ingersoll-Rand, Gardner Denver, Sullivan Machinery, Cleveland Rock Drill, and the Chicago Pneumatic Company.

The mounted drifter is used for horizontal development, adit and tunnel work, and in stoping where horizontal holes are an advantage. When mounted on a tripod or "wagon" it is also used in quarry and open-pit operations where bench heights are moderate, although in a few pits holes as deep as 40 ft. are drilled. Down-hole depth, however, is not in excess of 20 ft., and for toe-hole work 30 ft. is about the limit.

Drilling rate depends upon the depth and diameter of the hole and the hardness and uniformity of the rock. As an example, in a hard-rock (granite) tunnel, 9 by 11.25 ft. in section, automatic feed drills with a 3½-in. cylinder gave average drilling speed in feet per minute ranging from 1.88 to 2.43 for five drills mounted upon a jumbo, or 4.5 to 5.8 in. per min. per drill. Included in drilling time was the time required for moving in the jumbo, setting up the drills, and drilling the round. For 10-ft. holes 1⅞ to 1⅜ in. in diameter and $\frac{1}{16}$-in. bit changes, H. A. Walker gives drilling rates at the Homestake mine of 13.78 in. per min. in medium-hard porphyry and 6.65 in. per min. in very hard quartzite. H. Dewitt Smith and W. H. Sirdevan gave drilling rates at the United Verde mine of 9.2 ft. per drill shift in massive sulphides (5-ft. holes), 37 ft. in schist (5.5-ft. holes), and 44 ft. in porphyry (6-ft. holes). At the same mine stoping massive sulphides gave 24 ft. (5-ft. holes) and schist gave 48 ft. (6-ft. holes) per drill shift.

Hand-held Hammer Drills, 3.—Drills of this type (Fig. 38) are similar in construction to the drifters but are much lighter and are equipped with handles, although various rigs are also in use to obviate the necessity for holding the drill or to control it better. The weight classes are as follows: Light, 28.5 to 32 lb., length 17 to 21½ in., ⅞-in. steel; medium, 35 to 50 lb., length 18½ to 22 in., ⅞-in. steel; heavy, 55 to 58 lb., length 21½ to 24 in., ⅞- or 1-in. steel; and very heavy, 62 to 85 lb., length 24½ to 24 in., 1-in. steel. Still heavier hand-held drills weighing up to 132 lb., length 32¼ in., and using 1¼-in. round steel are used. They are drifters equipped with handles instead of shells for use in very heavy shaft-sinking service.

Drill-steel chucks are for collared, lugged, or straight steel for use with a tappet front end. Some of the lighter drills are equipped with extra-strong rotation on the forward stroke and combine auger with hammer action. Variations in design for dry, wet, or strong blowing are available. Control is by throttle and water valve; some drills also have a rotation release. Various designs of steel retainers are in use. Usually the retainer is a spring-supported yoke that surrounds the drill steel below the collar on three sides and can be swung back to release the steel. The hand-held drill is a down-hole driller but is often used for horizontal and above-horizontal holes. Light cradles with screw or pneumatic feed for mounting upon light columns are available for such service.

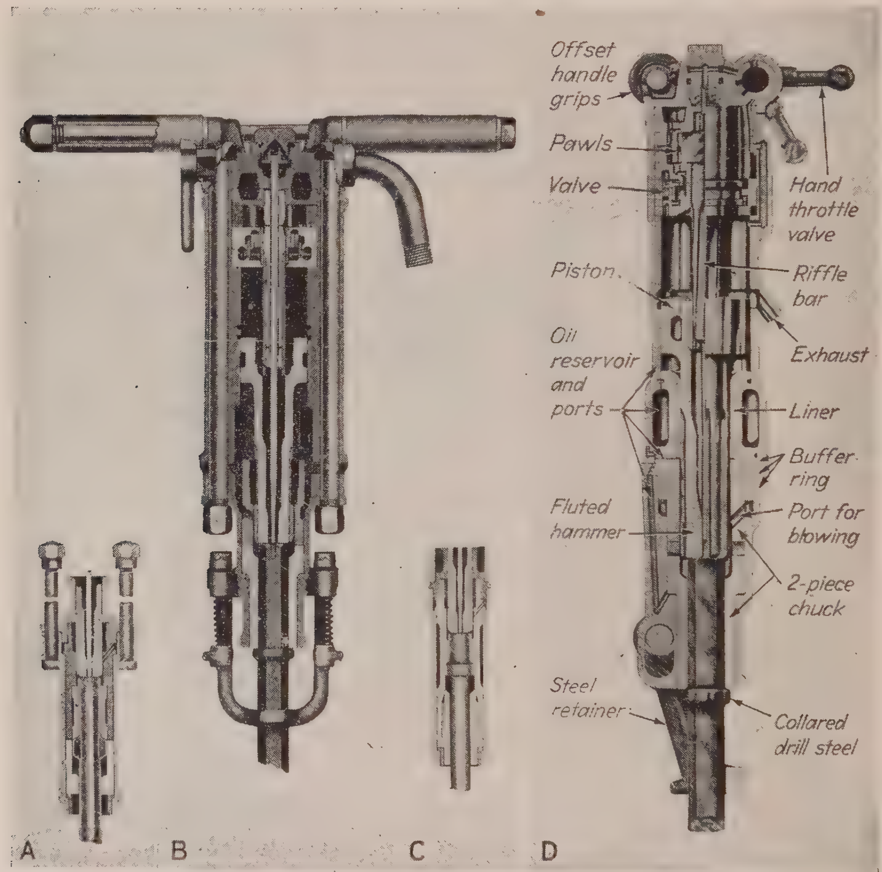

Fig. 38.—Hand-held drills in section—*A*, chuck for lugged steel, *B*, for collared steel, and *C*, for straight steel. (*Cleveland Rock Drill Co.*) *D* is an example of a heavy sinker. (*Sullivan Machinery Co.*)

This drill is of general utility in surface and underground operations. The heavier drills are used in shaft sinking; cradle-mounted drills are used in place of drifters in drifting and crosscutting; the lighter drills are used for blockholing, squaring up and taking up floor humps, hitch cutting, survey plug holes, and ditch, trench, and foundation work. Special types are used for sampling, spade, hammer and pick work and for sheet pile driving. It displaces many of the hand tools of the miner and is faster and more efficient. In open-pit mines and quarries the hand-held drill is used for primary blastholes 15 to 20 ft. in depth. For such work a light pipe tripod and block and tackle are often used to facilitate handling of the drill and long steel. Auger-type drills are used in iron-ore mining and in industrial mineral mining where the ore or mineral is not too hard.

Drilling rates for 12- to 14-ft. holes are 150 to 200 ft. per shift in shales and soft limestones and 60 to 120 ft. in harder rocks. A. E. Payne gives drilling rates for stoping in the Transvaal Rand mines as 45 to 300 ft. and an average of 75 ft. per shift for drill holes not exceeding 4 ft. in depth. W. A. Knoll and J. C. Sullivan in describing shaft sinking in granite on the Gogebic gave the following data: Shaft section 13 by 21 ft., depth of round 10.83 ft.; average depth of drill holes 12½ ft.; starter bits 2⅞ in.; finisher bit 1.5 in.; steel changes 12 in.; gage changes ⅛ in.; 850 ft. drilling per round, 68 holes per round, time 9.5 hr.; 12 hand-held drills, 70.8 ft. per drill shift; 7.45 ft. per hr., 1.5 in. per min. (3-in. piston, 76-lb. drill); 800 pieces of drill steel per round, breakage 50 to 100 pieces.

Stoper Hammer Drills, 4.—Stopers are hammer drills mounted in line with an air-feed cylinder. They are similar in construction to other hammer drills. Two principal kinds are made, the hand-rotated and the automatically rotated. Hand-rotated stopers are of either the wet or the dry type; self-rotated stopers are of the wet type. Throttle and feed control are provided, and most self-rotated stopers also have a rotation release.

Stopers (Fig. 39) range in weight from 77 to 125 lb.; closed length is from 51½ to 65¾ in.; cylinder diameters 2¾ to 3⅛ in.; feed lengths 18 to 24 in.; feed-cylinder diameters 2⅜ to 2¾ in. Front-end construction comprises chuck, tappet, chuck housing, and sometimes a shank aligner. Provision is made for keeping cuttings and water out of the drill by a constant blow port, and watertube construction prevents backflow into the drill mechanism. The hollow piston rod of the feed cylinder is provided with a steel point, which supports and maintains alignment of the drill. In the Sullivan stoper (4*c*) the drill cylinder is offset from the feed cylinder, which carries a pressure-operated upper rod and steel crowfoot to hold the stoper in a fixed position. Alignment is thus maintained without effort on the part of the drill operator. Separate throttle control is provided for the feed cylinder and the top rod. A feed length of 30 in. is furnished.

The stoper is used principally for uppers and holes at angles greater than 20 to 30 deg. above the horizontal. Its important field is in raising and back stoping where uppers are advantageous. H. Dewitt Smith and W. H. Sirdevan give some drilling rates at the United Verde mine: In massive sulphides 11 ft. per machine shift (4.3-ft. holes), in schist 28 ft. (6-ft. holes), and in quartz prophyry 48 ft. (8-ft. holes). These rates are for relatively hard rocks; for softer rocks drilling rates are often very high.

Rotary Drills, 5.—Cutting is accomplished by pressure and a relatively high-speed rotation of the bit or cutting head. The hand-held drill (5*a*) is operated by a compressed-air motor or an electric motor. The Jeffrey A-7, for example, weighs 40 lb. and has a 1½-hp. 250-volt d-c motor and a switch-controlled trigger or handle. The Joy drill weighs 43 lb. and has a 3-hp. 230-volt motor; the 5-ft. boring bar weighs 14 lb. and encloses an inner and an outer cutter revolving in opposite directions, with return of cuttings to a discharge close to the drill, where a bag can be used to catch the drill cuttings.

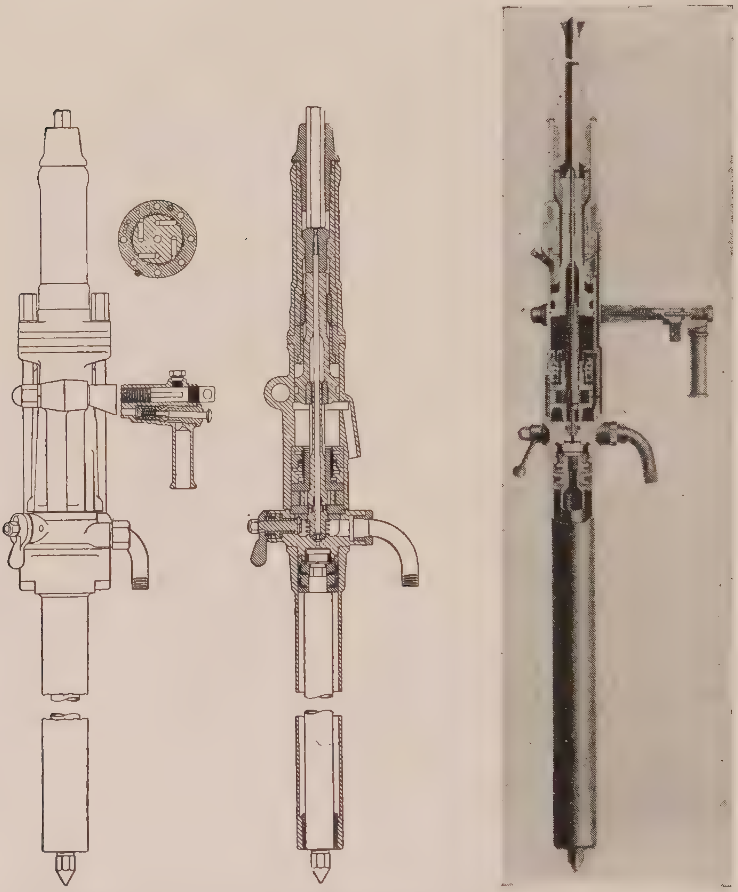

FIG. 39.—Sectional views of stoper drills—left pair (*Ingersoll-Rand Co.*); right (*Cleveland Rock Drill Co.*)

Mounted rotaries are driven by 250-volt d-c or 220-volt a-c reversible motors. The motor is placed parallel with the thread bar and drives it through a train of gears and splines, which are in keyways on opposite sides of the thread bar (in effect a feed screw). Forward motion is imparted to the tread bar by a stationary split nut (liner), which is closed over the thread bar by a safety housing on the forward end of the drill (a clutch arrangement is also used in some drills). Opening the liner stops forward motion, so that the thread bar can be pushed back. A socket provided with an outer sleeve is attached to the forward end of the thread bar; the socket holds the auger drill steel. Gear ratios are changeable, so that speeds of the thread bar may be varied from 99 to 228 r.p.m. for different drilling conditions. Motors are from 1.45 to 3.55 hp. for different drill sizes. Drill feed speeds are from 16 to 29 in. per min. in coal and 12 to 18 in. per min. in rock; gears must be changed to vary feed speeds. Motor assemblies weigh 94 to 101 lb.; a 5-ft. thread bar weighs 15 lb.; a 5-ft. column mounting weighs 75 lb. The column is provided with an offset top and an offset bottom jack, the turntable carriage upon which the drill is mounted being secured by a latch at any desired height upon the notched column. The foregoing description covers the Superior drill (Dooley Bros.). Cutters or bits are shown in Fig. 40.

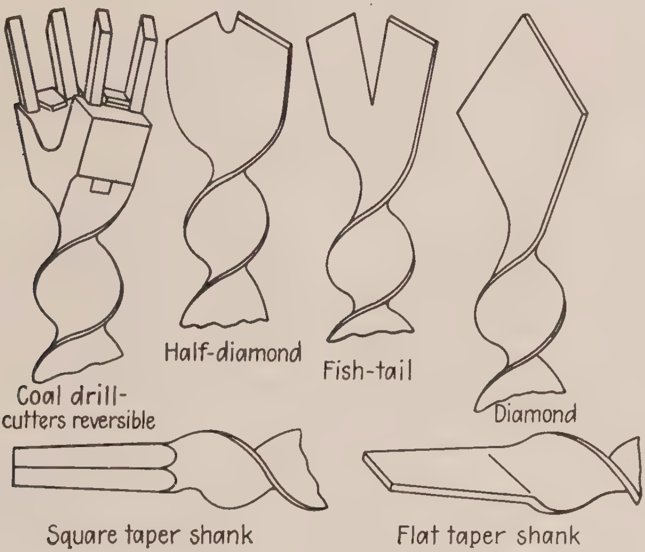

FIG. 40.—Rotary cutting bits and shank detail.

The Jeffrey car-mounted drill (5c) consists of a motor-propelled four-wheel car carrying two drills mounted on 7-ft. radius booms, which can each be swung over 90-deg. arcs from front to side; the drills can be swung in 180-deg. arcs of 3 ft. 11 in. radius in a vertical plane, enabling drill holes to be spotted in rooms 12 to 24 ft. in width. Drill motors are 3 hp.; weight of drill rig is 3.85 tons and travel speed 3½ to 6 m.p.h. The drilling machine is in effect a well-designed jumbo. The Sullivan automatic drill (5*d*) is mounted on a four-wheel rubber-tired truck, motor driven. The drill boom, carrying a 3-hp. rotary drill, can be swung so as to cover the drilling positions.

The horizontal strip borer (5*e*) is used in coal strip pits. Rotary drills of this type (Fig. 41) are usually driven by gasoline engines, but some designs make use of electric motors. The Sullivan strip borer is equipped with tread tractors; the drilling head can be swung in a vertical plane and carries hydraulic feed cylinders that exert a pressure up to 5 tons upon the kelly bar. The chuck end rotates the auger drill rods. The drilling head can be raised from a position 18 in. above ground level to a height of 90 in. Rotation is in two forward speeds, 84 and 160 r.p.m., respectively, and one reverse speed. Feed length is 30 in., but the kelly bar is notched at 30-in. intervals, providing four settings for each 10-ft. length of drill rod. The machine weighs 5.5 tons. Depth of drilling is 75 to 100 ft. with 6-in. diameter bits. The Parmanco side-wall driller is mounted on two rubber-tired wheels with the front of the frame supported by two vertical jacks. An inner sliding

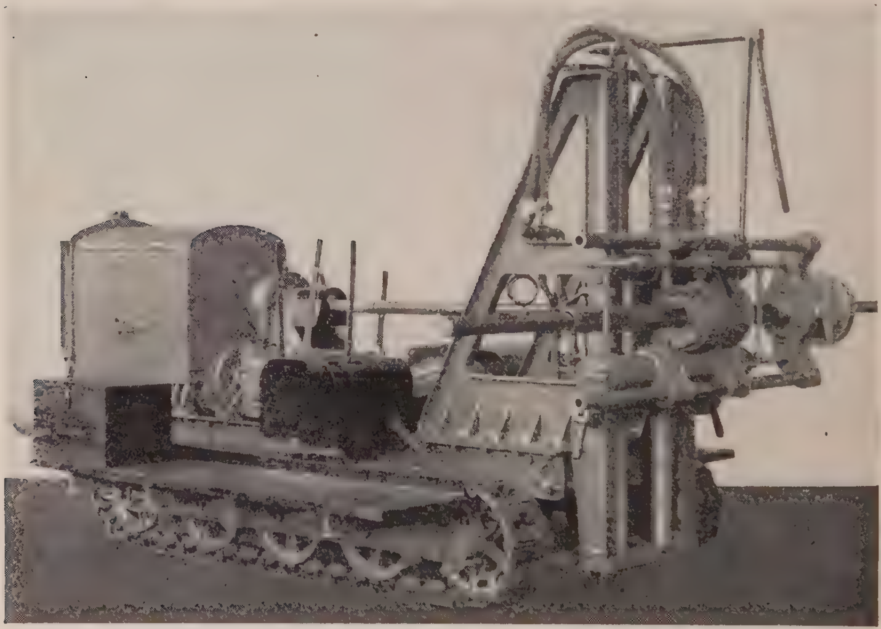

FIG. 41.—Gasoline-engine-powered stripborer. (*Sullivan Machinery Co.*)

frame carries the 16-hp. gasoline engine, the speed reducing unit, and the drill chuck. The rack and pinions, hand operated, thrust the frame forward as drilling advances. Auger speeds are 80 to 165 r.p.m. The machine weighs 1 ton.

The diamond drill (5*f*) has been described in the preceding chapter. All forms of rotary drills just described operate dry, in contrast to the diamond drill, which requires a stream of water through the drill rods to the cutting bit. Diamond drills are used for drilling long blastholes in stoping and pillar work. The availability of light drills for this work has expanded the field of the diamond drill, and it is now often included in underground drilling equipment.

The principal field of the rotary drill is in coal and industrial mineral mining, where the rocks encountered are not too hard. Salt, potash, clay, gypsum, tripoli, and similar minerals are examples. Holes are horizontal and range from 6 to 10 ft. in depth; they are 2 to 4 in. in diameter for coal and up to 2¾ in. for other minerals. Drilling rates are very high as compared to those obtained with hammer and churn drills, which are used upon much harder rocks. Thus drilling holes in clay with fishtail bits of 2¾-, 2½-, and 2¼-in. diameter gave 216 ft. of hole per shift. Drilling speeds of hand-held drills in coal are claimed to be up to 7 to 9 ft. per min.; mounted coal drills have feeding rates of from 18 to 29 in. per min.

Horizontal strip borers have a range of 60 to 75 ft. with hole diameters of 6 in. Drilling rates are high, but much depends upon the softness of the stratum within which the hole is drilled. Usually a shale seam is selected. One example gives a drilling rate of 46.5 ft. per hr. for holes 48 ft. deep.

The light diamond drill is a long-hole driller for hard and medium-hard rock. Considerable application has been made of this kind of drilling in sublevel stoping, pillar recovery, and to some extent in shrinkage stoping. In sublevel stoping the maximum depth of holes is about 75 ft.; in shrinkage stoping horizontal holes up to 135 ft. in depth have been used. Holes are 1.5 in. in diameter and may be drilled at all angles. The first application of diamond drills for blasthole purposes was probably at the Soudan mine (Minnesota), where exceptionally hard rock and ore were encountered. Probably the next application was for long-hole drilling in pillar mining. At Noranda in 1935 diamond drilling was found advantageous in sublevel stoping for holes up to 65 ft. Since that time other mines have made use of long diamond-drill holes. According to F. S. Dunn, drilling rates at Noranda ranged from 15 to 45 ft. per shift, with an average of 25 ft. At the Soudan mine, diamond drilling of blastholes was resumed in 1938 with holes 1.5 in. in diameter and up to 20 ft. in depth in ore. W. P. Wolff gives the following drilling rates: For 3.75- and 5-hp.

electric drills, elapsed time was 0.814 and 1.061 in. per min., respectively; actual drilling time, 1.243 and 1.76 in. per min.; inches drilled per bit, 14.21 and 21.59. In pillar operations at Copper Mountain, British Columbia, W. L. Nelson and F. Buckle employed 2.75-in. diamond-drill holes for both uppers and downs, 55 to 210 ft. in length, and obtained an average of 14 ft. of hole per shift on uppers.

Percussion Drilling.—In churn drilling, the string of tools is lifted and dropped. The bit strikes the ground at the end of its stroke, penetrating and rebounding. The rapidity of the fall, the weight of the drill string, the sharpness of the bit, and type of rock determine the penetration per blow. Successive blows cut into new positions; there is also a certain smashing effect as foot-pounds of energy are concentrated on a relatively small area of rock and momentum is absorbed. Because the string of tools is suspended upon a wire rope, the area cut by the bit exceeds the diameter of the bit by an inch or more. Churn-drill holes usually maintain their diameter. Too heavy a blow relative to rock resistance may result in sticking of the bit. It is for this reason that variation in strokes per minute and in length of stroke is provided. The skill of the drill runner determines the nice adjustment required for maximum rate of cutting without sticking. Table 11 gives a classification of rocks according to drilling character.

The piston of the hammer drill strikes a blow on the head of the drill rod or the tappet, which transfers the blow to the drill rod, which in turn transfers the blow to the drill bit pressed against the bottom of the hole. The sharp edges of the bit cut the rock, and the steel rebounds as the piston returns and the rotating mechanism turns the drill steel and bit

TABLE 11.—ROCK CLASSIFICATION

Hard	Intermediate	Soft
Granite	Rhyolite	Altered rocks
Aplite	Andesite	Serpentine
Syenite	Dacite	Gypsum
Diorite	Trachyte	Shales
Basalt	Phonolite	Rock salt
Dolerite	Glassy volcanics	Anhydrite
Silicified rocks	Tuffs	Iron ores
Flint	Sandstones	Sandstones
Quartz	Slates	Coal
Quartzite	Limestones	
Siliceous hematite	Marble	
Magnetite	Altered rocks	
Pyrrhotite	Iron ore	
Diabase	Dolomite	

to a new position. In contrast to the churn drill the force of the blow is small, but the number of blows, up to 3,000 per minute, is great. Though feed pressure keeps the drill bit against the bottom of the hole, sticking results from too heavy a blow for the type of rock, or where seams, fissures, and soft places occur. By means of the throttle and feed adjustment the number of blows and their intensity may be varied to suit the conditions.

A succession of bit sizes is necessary on account of dulling cutting edges and reaming wear upon the sides of the bit. Holes are largest at the top and smallest at the bottom, the abrasiveness of the rock drilled determining reaming wear and the hardness and toughness of the rock the rapidity of dulling. Drill changes range from 12 to 24 in. or even more; gage changes of ⅛ in. are common, but precision in sharpening bits has made possible a $\frac{1}{16}$-in. gage reduction in successive sizes where reaming wear is not too great. Drill holes are finished at 1⅜-in. diameter for a cartridge size of 1¼ in.; this establishes the starting gage when the number of drill changes for a given depth of hole is known. Drill changes and gage changes are standardized for conditions in a given mine.

The smaller the starting diameter, the greater the rate of drilling, other things being equal; drilling speed varies inversely as the square of the diameter of the holes or inversely as the volume of rock removed. Hardness, toughness, the grain size of the rock, and the nature of the cementing material are factors influencing drilling rates. Although efforts have been made to work out a relation between drilling rates and rock characteristics, nothing of any great accuracy has been established; the best measure is trial and observation of drilling for rock variations under working conditions in a mine.

With a given diameter of drill hole, the greater the intensity of the blow, the deeper the penetration; and the greater the number of blows per minute, the more rapid the drilling rate. Higher air pressures with a given size of drill usually give greater blow intensity; with the same air pressure and a larger cylinder diameter of the drill, a greater intensity of blow can be obtained. However, steel breakage enters as a limiting factor. Modern drills are designed and air pressures are limited to a range that will avoid excessive steel breakage. Dull bits also result in steel breakage as does too small a steel section for the drill size. Balanced equipment and plenty of sharp steel give less broken steel and lower machine maintenance. The human element is of great importance. Experienced drill runners are necessary with modern drills.

Drill Operation.—Compressed-air drills operate at 80 to 90 lb. per sq. in. with compressor pressures 90 to 100 lb. per sq. in. Pipe installation requires good design and appropriate selection of main, intermediate, and working-place pipe sizes. Receivers are placed at intermediate points where long pipe lines are necessary. Probable peak demands for compressed air are provided for various working places. Drill lubrication is important, not only the type of lubricant but its systematic use. Rubber hose of good quality connects pipe line and drill, and line oilers are interposed close to the drill for delivering oil to the air stream. Most drills are equipped with oil plugs at critical points. As each type of drill has its operating characteristics, training is necessary to obtain operators who will get full performance with a minimum of maintenance and repairs to equipment.

Although uppers clean themselves of drill dust, it has nevertheless proved to be a menace to health, and wet drilling is necessary to avoid it as much as possible. Usually water lines are established to bring water to the drills. Where this cannot be done, water tanks of 6 to 15 gal. capacity with hose connections for compressed air and for delivery of water to the drills are maintained in the working places. Down holes are cleaned by blowing with compressed air delivered either through the drill or by a blowpipe attached to a separate hose. Many drills have special valves for blowing holes.

Light drifters are operated by one runner; heavy machines have a helper as well, although he may serve two or more drills. In high-speed tunnel work two men may be used for each drill at the face. Tool carriers or nippers distribute steel or drill rods to working places. The use of detachable bits has greatly lessened tool carrying.

Bringing drill and equipment to the face and setting up machines require an appreciable time, which reduces the actual drilling time per shift. Likewise the tearing down of the drill consumes time. Over-all drilling rates include total time from bringing in equipment to its removal; actual drilling time rates include only that portion of the time involved in drilling, changing steel, and shifting the position of the drill to a new hole. Usually for each setting up of the column there is a round of holes that is drilled from the setup. In narrow drifts one machine drills the round; in a wide drift two machines may be mounted upon crossarms, each drilling half of the round; in large-section tunnels four or more drills may be operated from a jumbo.

The primary purpose of drill holes is to provide for the placement of explosives in such amounts and at such places as will yield minimum cost in the displacement of ore and rock. Drilling efficiency is attained when over-all blasting costs are lowest.

Drill Steel.—Weights and dimensions of drill steel are given in Tables 12 and 13. Cruciform steel is now uncommon. Hollow drill steel in round, hexagon, and quarter-octagon sections is widely used, although solid steel is sometimes used for starters. High-carbon steel (0.90 per cent) is favored for forged bits, but the widespread use of detachable bits has resulted in the selection of more suitable steels for drill rods. Three shank designs are in use: The straight shank in quarter octagon or hexagon, the lugged shank for round, and the collared shank for hexagon and quarter-octagon steel. Shank ends are hardened in oil and ground square to prevent cupping or injury to the drill piston. Straight shanks require the use of a tappet or anvil between the piston and the shank end.

The moderate fatigue resistance of conventional drill steel is lowered by continued use of the same drill, so that the rate of steel breakage increases with the life of the steel or the number of uses. The replacement of broken steel by new establishes after a time an average condition, from which an average steel breakage may be ascertained. The quality of the drill steel is an important factor. G. R. Heywood made impact tests of different brands under South African stoping conditions, in which drills were made up and operated until broken. Average footages for different brands ranged from 93 to 151 ft. of hole drilled; mine tests gave an indicated life of about 138 ft. for a single drill steel of the best brand. C. G. Kemsley gives comparative results of an alloy steel (carbon 1 per cent, chromium 1.26, vanadium 0.1, manganese 0.32, molybdenum 0.3), ⅞-in. quarter octagon, in replacing 1-in. quarter-octagon carbon steel at the Dome mines. Steel breakage was reduced from an average of 2.9 per cent for the carbon steel to 0.445 per cent for the alloy steel.

TABLE 12.—WEIGHT OF SOLID DRILL STEEL, POUNDS PER FOOT[a]

Size, inches	Cruciform			Round	Octagon	Hexagon	Quarter octagon
	Light	Rocky Mountain	Heavy				
⅞	....	1.70	....	2.04	2.16	2.25	2.45
1	2.17	2.20	2.49	2.67	2.82	2.94	3.20
1⅛	2.76	2.80	3.10	3.38	3.57	3.73	4.05
1¼	3.36	3.35	3.61	4.17	4.40	4.60	5.05

TABLE 13.—WEIGHT OF HOLLOW DRILL STEEL, POUNDS PER FOOT[a]

Size, inches	Size of hole, inches	Round	Hexagon	Quarter octagon
⅞	5/16	....	2.00	2.19
1	¼	2.50	2.70	3.03
1⅛	5/16	3.14	3.46	3.79
1¼	⅜	3.81	4.11	4.64

[a] From "Colonial Rock Drill Steel Book," issued by Colonial Steel Co., Pittsburgh, Pa. Unit 0.283 lb. per cu. in.

Long-hole drilling is facilitated by the use of sectional steel. The maximum length of drill steel in one piece is from 18 to 24 ft. Although steel of this length is used in some mines, such lengths are often inconvenient to use underground. They are used in open-pit operations, however. In the preceding chapter long-hole drilling with sectionalized steel was described, but there the purpose of the drilling was for prospecting. At Noranda sectionalized drill steel is used for blasthole drilling in stoping operations. It is made up in 24-in. to 18-ft. changes, the longest piece being 6 ft. Shanks and filler sections were 1⅛-in. round nickel-chrome steel forged with a 1-in. quarter-octagon shank for the drill chuck; bit sections were 1⅛-in. high-carbon steel; couplings (Fig. 42) were of heat-treated chrome-vanadium steel, 2¼ in. in length by 1½ in. o.d., five rounded threads per inch; bits were four-point double taper with starting gage 2¹⁄₁₆ in. and finishing size 1⁵⁄₁₆ in.

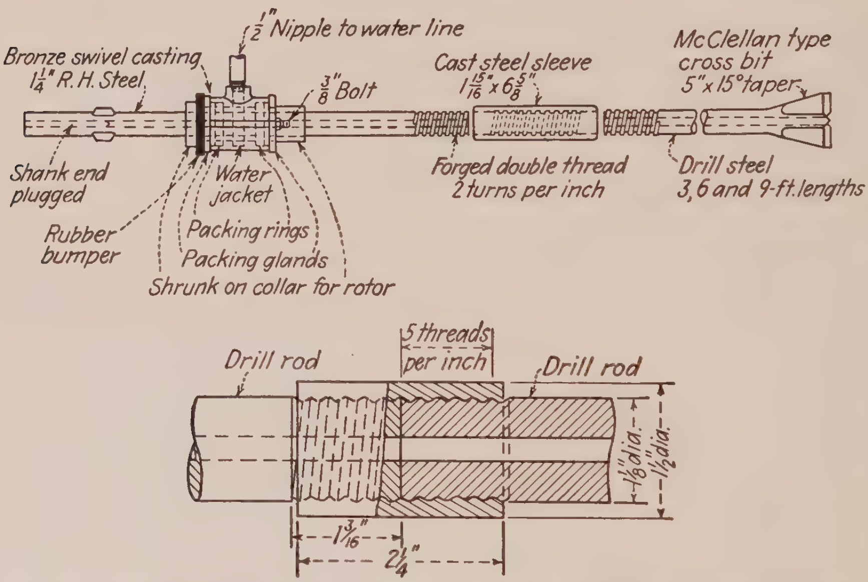

FIG. 42.—Method of coupling sections of drill steel for long-hole drilling. (*C. A. Dobbel.*) Lower drawing is the method used at the Noranda mine.

Drill Bits.—Drill bits forged integral with the rod were standard for many years. A. L. Hawkesworth applied for patents for a detachable bit in 1918 and received patents in 1922–23. Anaconda tried out and adopted the Hawkesworth bit for its Butte mines and thus gave impetus to the use of detachables. They obviate a considerable portion of steel handling and permit of the use of a better steel for the bit and more thorough heat treatment of the bit. A high percentage of mines now use detachables in the greater part of their mining operations.

There are several types of detachables (Fig. 43). The Hawkesworth is now obsolete. The screw-thread design appears to be in general use,

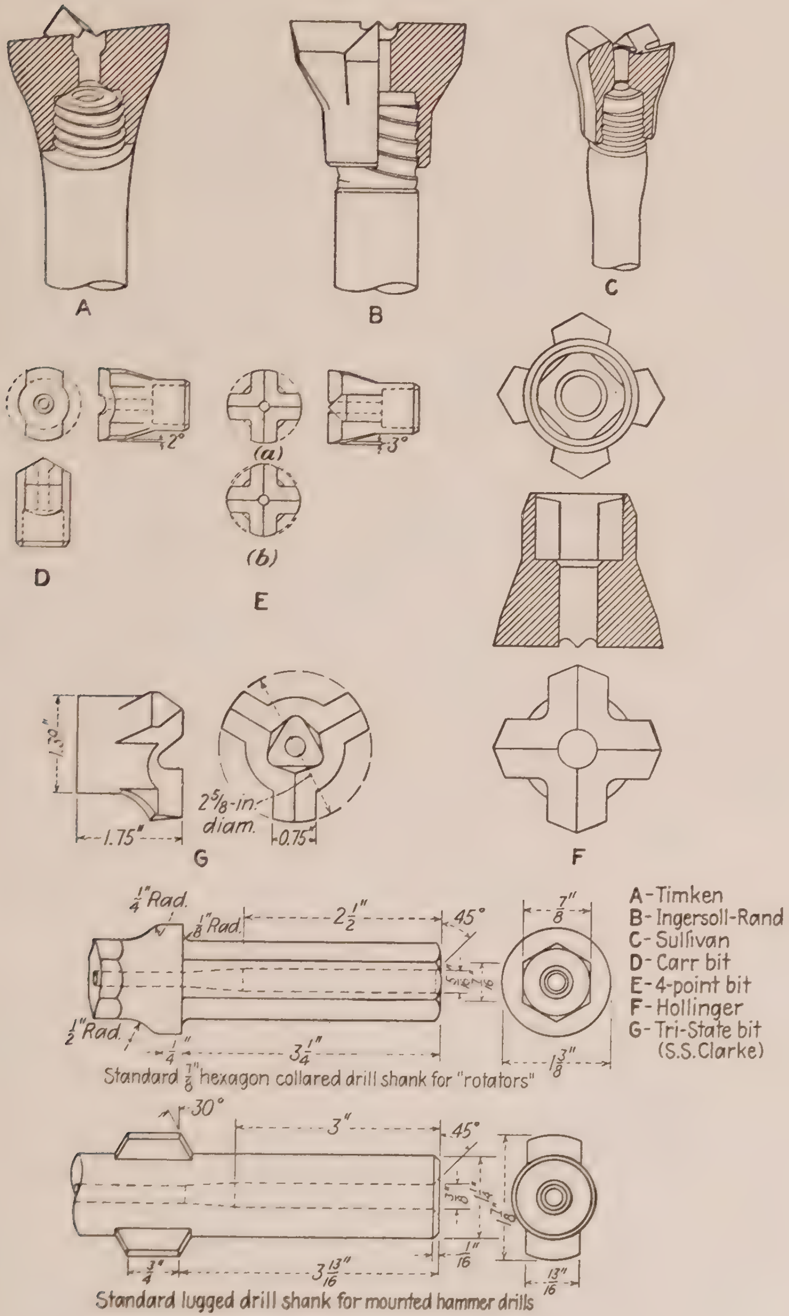

Fig. 43.—Detachable drill bits—standard shanks.

although there are detachables of slotted design. The Timken bit is screwed upon the upset end of the drill rod and bears upon a shoulder turned at the base of the thread, which is rounded. The Ingersoll-Rand Jackbit bears squarely upon the end of the drill rod and employs a thread relatively shallow and of wedge-shaped section. Sullivan detachables make use of a special acme thread that gives a bearing upon the end of the drill rod; an alternate design, similar to the Timken, makes use of a threaded upset that gives a bearing for the skirt of the detachable. The Liddicoat bit, used at Hollinger, is of a socket type; the drill rod is rounded with four surfaces milled upon it to correspond with four similar surfaces within the socket of the bit. The bit socket is a driving fit upon the end of the drill rod, and the milled surfaces prevent turning of the bit; in addition the skirt of the bit is annealed and is driven behind four undercut lips on the rod, crimping the skirt to the rod and giving an added hold on the bit when it is withdrawn (A. H. Wohlrab).

The shape and design of the bit present many variations. However, practice indicates three important designs, the four-point (cross bit), the six-point or rose, and the chisel or Carr bit. The center-hole type is generally used, but all designs are also made with side hole for use in clayey, seamy ground or where trouble is experienced with center-hole plugging.

TABLE 14.—COMPARATIVE BIT SIZES, HOMESTAKE MINE
(*H. A. Walker*)

Diameter of bit	Starters	Drill length, ft.							
		3	4	5	6	7	8	9	10
A	2⅜	2¼	2⅛	2	1⅞	1¾	1⅝	1½	1⅜
B	1⅞	1¹³⁄₁₆	1¾	1¹¹⁄₁₆	1⅝	1⁹⁄₁₆	1½	1⁷⁄₁₆	1⅜

A = before Aug. 1, 1939; *B* = after Aug. 1, 1939.

Gage reduction varies from ⅛ in. to ¹⁄₁₆ in. per drill change. Table 14 illustrates comparative sizes. Improved shop practice has resulted in an over-all accuracy in bit preparation that makes possible ¹⁄₁₆-in. gage changes, and there is some opinion that even smaller bit changes are feasible. The usual double-taper cross bit has a 5-deg. angle away from the cutting edge and a 15-deg. taper to the skirt of the bit; its wings are ⅝ in. thick, its cutting angle 90 deg. Larger diameter bits have wings up to ¾ in. thick; in some cases the cutting angle is increased to 105 deg. and even more. The 5-deg. reaming clearance is reduced to 2 deg. and sometimes omitted with improvement in gage maintenance. The six-point bit follows the same general design. It is used for blockholing, for starters, or for fitchery ground. The Carr or chisel bit has a single cutting edge and a reaming clearance angle of 2 deg. It maintains its gage and in hard ground often gives better performance than the cross bit. The Sullivan detachable has one pair of wings of the usual kind and another pair with the cutting edges coincident with the periphery of the drill hole. The reaming angle is small. S. S. Clarke describes a three-point detachable (Tri-State district) that has given good drilling results in cherty beds where high abrasive wear results.

Detachables are made from high-carbon (0.80 to 0.90) and alloy steels. A fine-grained electric furnace steel is widely used. The general opinion is that alloy steel gives the best results. When detachables lose gage they are reground to the next smaller gage and resharpened. Several regrinds are to be expected. By reforging, hot milling, and hardening, the number of reuses is increased. C. A. Kumke states that a maximum of 10 reuses has been obtained. Cracked skirts, ruined threads, and broken points prevent reconditioning.

Drill rods for the use of detachables are made up in sets with length changes of 18 to 24 in. Starters are from 30 to 38 in. in length, while seconds, thirds, fourths, and fifths correspond to feed changes found applicable at any one mine. Rods are made from steel with carbon content ranging from 0.75 to 0.85 per cent. However, alloy steel can also be used; what is wanted is a steel of high fatigue resistance. Threaded ends are hardened in oil and drawn, and chuck ends are also hardened in oil but not drawn.

C. L. Berrien estimated that the continuous operation of 85 working places for one year at Butte required an average installation of 4,020 pieces of steel averaging 15.29 lb. each, or a total of 61,479 lb.; 148,862 pieces of drill steel were sharpened during the year; the total weight of sharp steel handled was 1,137 tons (one way). With detachables the total weight of rods and bits handled was 219 tons (one way). A. H. Wohlrab gives the following data for detachable bit operation at Hollinger on a per-machine-shift basis: Rods damaged or broken, 0.86; bits used, 41; bits lost, 0.13; bits damaged, 0.21; holes drilled, 18.5; feet drilled, 114. Table 15 gives comparative results of detachables and conventional steel.

TABLE 15.—DETACHABLES AND FORGED BITS, TRI-STATE
(C. W. Nicolson)

Bits	Holes drilled	Ft. drilled	Ft. per bit	Bits dulled	Steel broken	Ft. per break
Detachable.	33,012	292,887	18.15	16,136	2,511	116.6
Forged.	4,373	36,644	4.30	8,570	676	54.2

NOTE.—Comparative costs: Forged bits—per bit dulled, 16.95 cts.; per foot drilled, 4.94 cts. Detachable bits—standard bits, 4.71 cts. per foot drilled; A-2 detachable, 3.01 cts. per foot drilled.

Drill-rod breakage (new Swedish steel): Feet drilled per broken steel 891.3 at beginning of period and 266.3 ft. at end of six months; same drill rods.

Distribution of Steel.—In small mines, drill steel is transported in mine cars and distributed to the working places by the nipper or miner. Special cars for drill-steel transportation are generally used at large properties and its distribution is methodical. Stocks of sharpened steel are maintained on working levels. Dull and damaged steel is gathered after each shift and transported to the surface and delivered to the sharpening shop. As has already been shown, the weight of steel to and from the mine is greatly reduced when detachable bits are used. Detachables are placed in carriers that contain a full complement for a shift's operation and are given to the drill runner as he goes on shift. At the end of the shift the runner returns the carrier with used and unused bits. Drill-rod replacements are required, and for this purpose stocks of drill-rod sets are maintained at working levels and near working places.

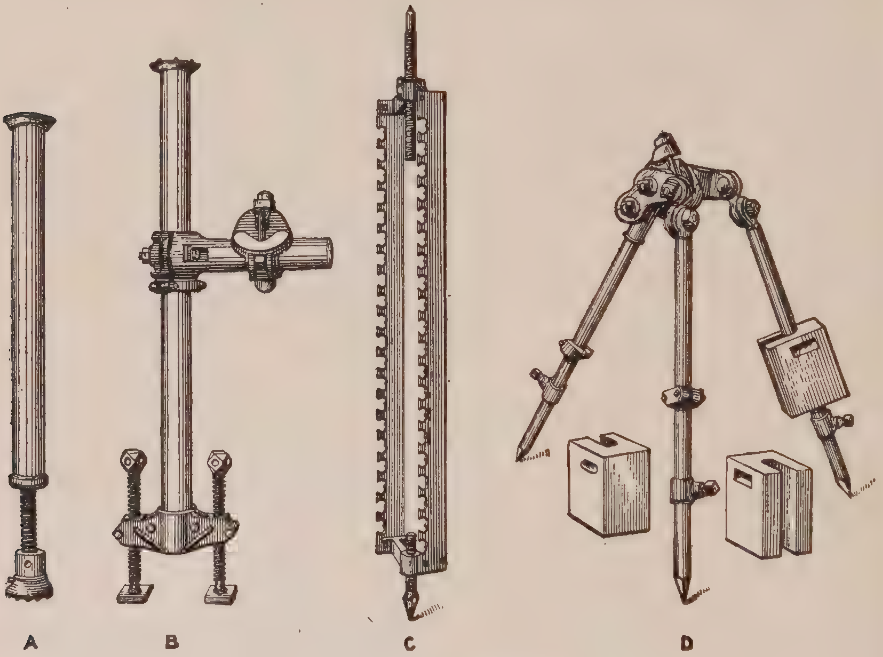

Fig. 44.—Drill mountings.

Drill Mountings.—Drifters and light mounted drills are supported upon arms attached to columns (Fig. 44) 2½, 3, 3½, or 4 in. in diameter. Column bases are equipped with either a single-or double-screw jack (the latter seldom used) at the base and a cap at the top. They are blocked top and bottom and held rigidly in place by the pressure of the jacks. Columns are made in various lengths; column arms are from 18 to 22 in. long; double-extension arms are of greater length. Safety clamps are placed below the arm clamp; the drill saddle clamp is attached to the arm. The arrangement is such that when the column is placed in position the arm may be shifted up or down and the drill in or out upon the arm, thus providing for changes of the drill position relative to the face. Extra-length columns are used for high stopes. Seamless-steel tubing, extra-heavy pipe, and aluminum alloy are materials used in columns.

Light hand-held drill mountings (fixed cone shell) for use upon light columns weigh 52 to 61 lb. and have a 26-in. feed length. A pneumatic leg support (Ingersoll-Rand Jackleg) weighing 35 lb., with bayonet-type attachment and 36-in. feed travel, enables the hand-held drill to be more conveniently used for horizontal and angle holes. The so-called "Mexican setup" consists of a 1-in. pointed steel bar with a forged collar 3 in. below the top; the drill is provided with a hook or ring, which is slipped over the end of the bar. Another form of pneumatic feed leg is attached to the drill by means of U bolts and enables it to be used for uppers. At Mount Hope, F. M. Radel made use of a stoper leg with special air connections attached to a hand-held drill. The combination was used in stoping. A 2 by 10 in. by 12 ft. plank was supported against the face at one end and the other end was wedged into the muck pile; then a line of holes was drilled in the plank for varying the position of a ring bolt that held the point of the stoper leg of the plank-supported drill. Holes up to 16 ft. in depth, from horizontal to 30 deg., were drilled with this rig in stope faces. Short ladders were also used for supporting the plank.

Steel tripods with leg weights of 220 to 560 lb. are used in open-pit work. Tripods are also used in open stopes. The quarry bar, 10 to 12 ft. in length, with saddle clamp and rack and pinion for horizontal travel, is used in dimension-stone quarrying.

Jumbos of many different designs are used in large-section crosscut and tunnel work. A jumbo consists of a four- or eight-wheel car carrying an assemblage of columns and arms for the attachment and positioning of the drills. The smallest jumbo is designed to support two drifter drills and the larger ones from five to eight or more. An intermediate platform, hinged, provides a working staging for the drills serving the upper part of the section. Compressed-air and water manifolds are included, as well as tool racks for drill rods. The purpose of the jumbo is to enable the drills to be moved into the face and to be placed in operation with a minimum of delay and, on the completion of drilling and loading of holes, to be removed away rapidly along with all tools and other equipment. Cleveland manufactures a two- or three-arm rig (Fig. 45) mounted on a four-wheel car. Each boom or arm is mounted so as to swing about a short vertical axis and to be raised or lowered in a vertical arc by means of a crank-operated worm and screw and to be locked into position; saddle pieces to which the drills are attached can also be swung. The boom arrangement offers ample coverage of drilling positions. Two compressed-air-operated jacks reacting against the roof hold the rig in position. Manifold and tool racks are provided. Gardner-Denver has an interesting drill-carriage design.

Wagon drill mountings (Fig. 46) are employed in open-pit and quarry operations. A great many designs are in use. One of the lightest, the Sullivan, has a two-wheel rubber-tired steel frame that carries the drill shell and is held in place by steel anchor points. It is in effect a wheeled tripod and weighs 375 lb. A heavier type weighing 1,475 lb. is an adjustable steel frame supported on three rubber-tired wheels. The steel change is 6 ft., the feed travel 9 ft. Feed is by chain operated by a compressed-air engine. A drill aligner is placed at the front of the drill shell for starting holes. The front wheels are equipped with brakes and can be swung on swivels at right angles for line drilling or drilling close to a vertical face; the rear wheel can be swung about its vertical axis. A hydraulic pump is provided for adjusting the frame; holes can be drilled from vertical to horizontal. Chicago Pneumatic manufactures a three-wheel mounting (steel wheels), providing 10- to 15-ft. steel changes, with air-operated hoist and tilting tower weighing 2,465 lb. with a 4-in. drill. The Ingersoll-Rand wagon drill is a 3-wheel steel frame (rubber tires)

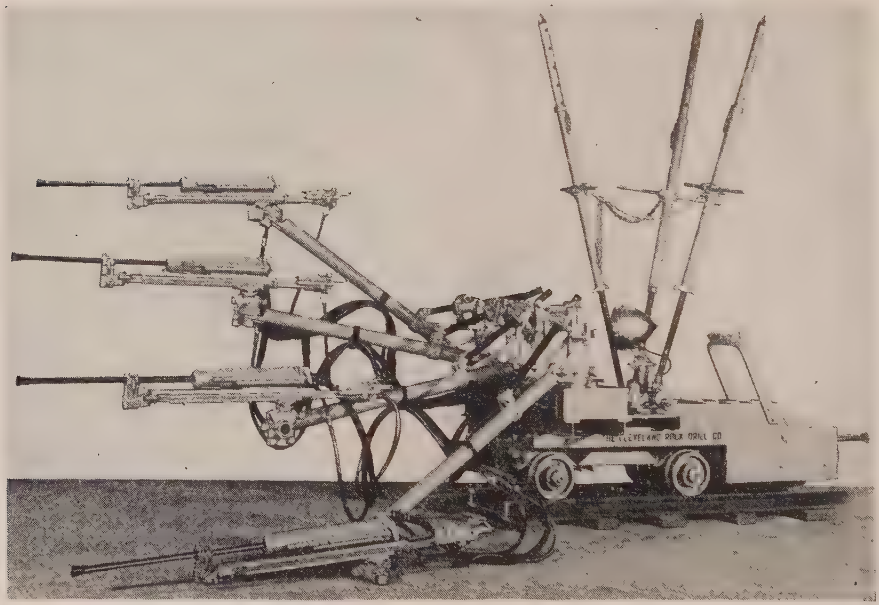

Fig. 45.—Drill carriage or jumbo. (*Cleveland Rock Drill Co.*)

with tilting feed frame and saddle that can be raised and lowered by a worm gear and a ratchet handle on two uprights; an air motor provides for feed and quick return; steel change is 6 ft., and a drill centralizer provides for starting holes.

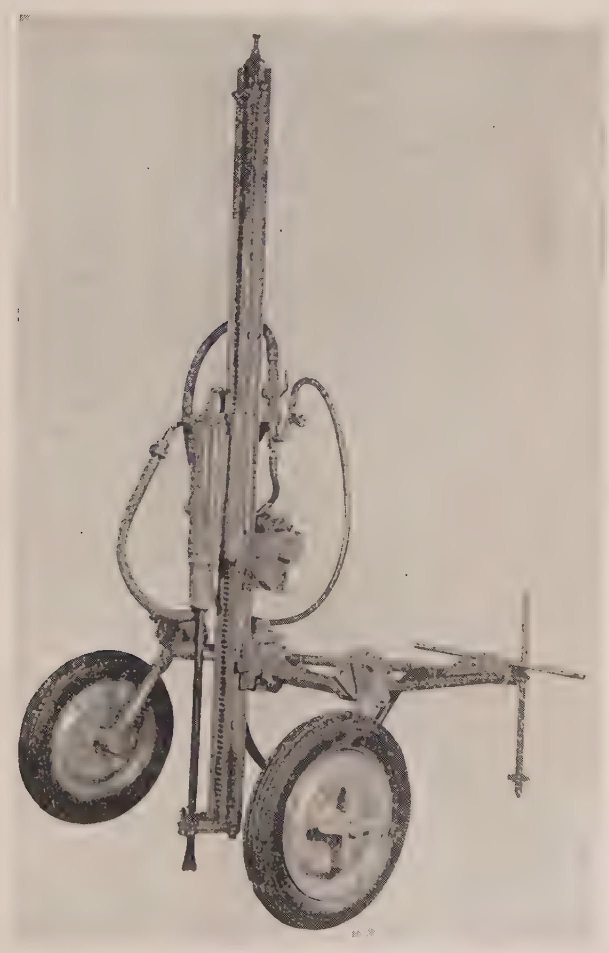

FIG. 46.—Wagon drill. (*Sullivan Machinery Co.*)

Future of the Rock Drill.—Piston rock drills were invented in 1849. Simon Ingersoll greatly improved early types in 1871 and thereafter. J. George Leyner patented the hammer-type drill with hollow drill steel in 1897. In 1905 the Waugh hand-rotated stoper appeared. The hand-held drill was introduced in 1912 by Ingersoll-Rand. Power-feed drills appeared about 1936. In a highly competitive field the design of rock drills has been steadily improved, and the availability of advantageous alloy steels has made possible great reductions in weight and dimensions. Modern rock drills fully meet present-day mining conditions. Without doubt the compressed-air drill will continue to be used for blasthole drilling in rocks from intermediate to hard, which covers most of the metal-mining conditions. For holes 20 ft. or more in depth the diamond drill possesses important advantages. Whether it will be used for short-hole work, except in the hardest rocks, is doubtful. The electric diamond drill uses power more efficiently than the air drill, but it also requires a considerable amount of drilling water, which is often difficult to procure.

Removal of Cuttings.—Hollow steel used with an internal jet of air and water facilitates the removal of cuttings. Holes at an angle above the horizontal present no special problem. The blow tube is used at intervals to blow the sludge out of down holes. Where dry drills are used the disseminated drill dust becomes a hazard. Even in wet drilling there is some dust. However, wet drilling and wet drills have displaced the dry drill in most underground operations.

Wet Drilling.—Dissemination of drill dust is at a maximum when a hole is being collared. Dust suspensions are greatest in dead ends, where ventilation may be at the minimum. Raising is probably the worst situation. Many efforts have been made to reduce dust as much as possible or even to eliminate it. Sprays of many kinds have been tried and discarded. Three methods of controlling drill dust have survived, but there is still opportunity for the inventor. These methods are the use of the wet drill, the provision of a sufficient volume of ventilation to abate the dissemination, and the wearing of respirators by drill runners. None gives a complete solution, and often all are used simultaneously. J. A. Johnson and W. G. Agnew experimented with a closely fitting spray ring that was slipped over the drill steel and rested upon the rock at the collar of the hole; spray holes were in the inner surface of the ring and the water was discharged about the drill steel. It was found effective in starting down holes.

Exposure to drill dust is least for power-feed drifters, independently supported stopers, and wagon drills, and greatest for hand-held drills, all of the wet type. The use of deflectors or shields to deflect the dust blast from the drill runner has not been tried as far as is known, although on some drills the principal part of the exhaust is from ports that are directed forward toward the drill holes and thus no doubt keep some of the dust away from the runner. Dust bags have also been tried with some success.

The wet drill contains a water tube that passes through the axial line of the drill mechanism and terminates within or at the hole in the drill steel. The tube is connected by inner passages to an outside valve and a ½-in. water hose. Water tubes vary in length from 12 to 27 in., depending on the size and design of the drill. Delaware Aqueduct standards for use of wet drills are in part as follows: Drifter drills (high air flow and low water flow), air through drill steel 4 to 6 cu. ft. per min., water 0.5 to 0.75 gal. per min. at full-speed operation, ventilation 2,500 cu. ft. per min. per machine. Drifter drills (low air flow and high water flow), air 0 to 2 cu. ft. per min., water 1.5 gal. per min. and over, ventilation 1,200 cu. ft. per min. per machine. Drifter drills (medium water and air flow), air 2 to 5 cu. ft. per min., water 0.75 to 1.5 gal. per min., ventilation 2,000 cu. ft. per min. per machine. Hand-held drills (maximum piston bore 3 in.), minimum water flow 0.75 gal. per min., an air flow at full speed 2 cu. ft. per min. (maximum) in rock formations containing 10 per cent or more of free silica, with a minimum of 500 cu. ft. per min. of additional mechanical ventilation at a working place in a confined area. The foregoing standards represent an attempt to define the relative flows of air and water through the drill steel and to indicate the additional amount of ventilating current required to abate the dust dissemination. They are of use because they give specific measures.

The diamond drill is a wet drill and produces no dust, as the stream of water is in relatively large volume and the cuttings are removed in the form of a dilute sludge. All other forms of rotary drills are dry drillers but remove their cuttings in comparatively coarse condition, and no hygienic objections have been raised to their use. In the Joy rotary drill a dust bag can be used, but this is for the purpose of keeping the dust out of the coal when drilling rock.

Drilling Rate.—Some examples of drilling rates have been given elsewhere, but comparisons require that all pertinent factors be given. The toughness, hardness, and crystalline structure of the rocks influence the rate; but sharp steel, well-maintained drills, and experienced runners are essential. The size and type of the drill and its support are also important. In each mine, drilling rates should be determined for the principal rock types encountered and for the sizes and kind of drills in use.

Drill-steel Changes.—Sets of drill steel are arranged in length increments of 1 to 2 ft. and longer where a feed-screw length of 2½ ft. is used. Tests should be made to determine the maximum and average distances drilled by standard bits in the different kinds of rocks in a mine. These data will enable standard drill-steel sets to be made for holes of various depths. For hard, tough rocks the change increment is usually 1 ft. and for intermediate rocks, 2 ft. The fewer the steel changes, the greater the time saved in drilling a hole of given depth. It is evident that for the shorter increment the number of drill steels per hole is doubled, thus requiring a greater time in changing steel, more time in getting steel to and from the working, and a greater amount of blacksmithing.

Diameter of Drill Holes.—Underground drilling employs holes of relatively small diameter. Former practice required starting diameters of 2 to 2½ in. and finishing diameters of 1⅜ in. Figure 47 shows dimensions of bits for ⅛- and 1/16-in. gage reductions. Nowadays, with better bit dressing and more accurate control of sizes, starters of 1⅞ to 2 in. are used. Holes drill slightly larger than these diameters. In South African gold mines where stoping holes are 3½ to 4 ft. in depth, starters are 1¼ and seconds 1⅛ in., using ⅞-in. steel. In coal mining, cartridge sizes are 1⅛ to 2 in. in diameter and auger drills 1½ to 2⅛ in. For Cardox and like shells, holes are 1¾ to 3 9/16 in. in diameter (finishing); for the du Pont hydraulic cartridge, 4¾ in.

Churn-drill holes are designated by the size of the bit used as 3, 4, 6, 6⅝, 8, 9, and 12 in. While churn-drill bits suffer considerable wear, their suspension on a wire rope results in a hole an inch or more greater in diameter than the bit size. There is a marked trend toward the use of larger diameter churn-drill holes, as these permit wider spacing and therefore fewer holes to break a given length of bank.

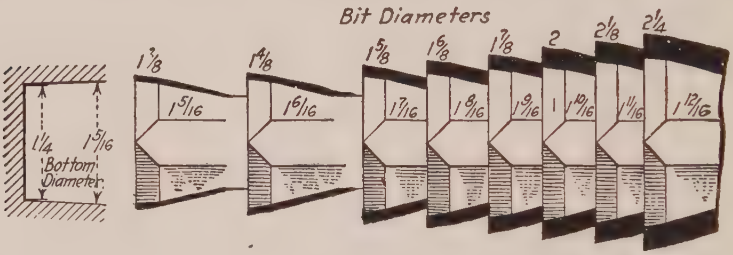

FIG. 47.—A set of drill steels. The inner set shown decreases in successive sizes by 1/16 in.; the outer set shown in black decreases by ⅛ in. for each successive size.

Drill Footage per Shift.—Actual drilling time in an 8-hr. shift ranges from 3 to 4 hr. Setting up a drill upon a column requires from 30 to 40 min.; "tearing down" somewhat less; loading holes 1 hr., and barring down at the beginning of a shift from 30 min. to 1 hr. One man is required for operation of the drifter, stoper, or plugger. In some instances he is given assistance in setting up the column in drifting work and in stoping where the drifter is used. Where light drill mountings are used assistance is usually unnecessary. Drill steel is distributed by tool carriers or nippers. Efficient tool and powder distribution increases effective drilling time, and cooperation on the part of the mucker facilitates setting up the drill. Figures 48 and 49 indicate aggregate theoretical footage drilled by drifter and plugger for holes of various depths and different drilling rates. For a given drilling time with all types of drills, the deeper the drill hole the greater the aggregate footage, because of fewer changes of the position of the drill. Faster drilling rates consequently give greater aggregate footages. Stopers and pluggers give greater footages than drifters, as less time is required for shifting and steel changing.

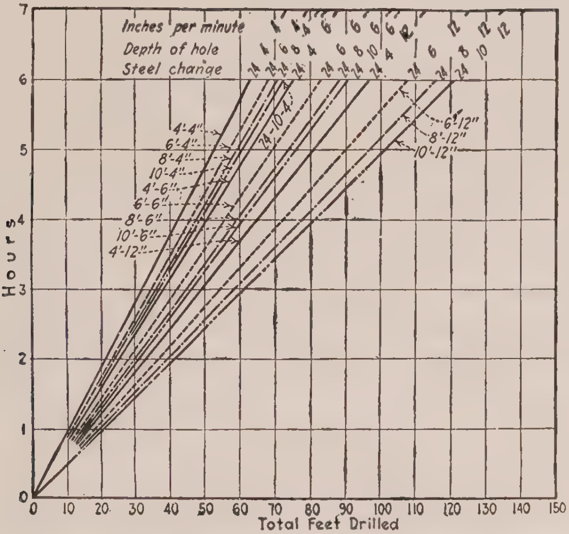

FIG. 48.—Estimated footage drilled with drifter—assuming 24-in. steel change, holes 4, 6, 8, and 10 ft. in depth, and drilling rates of 4, 6, and 12 in. per min.

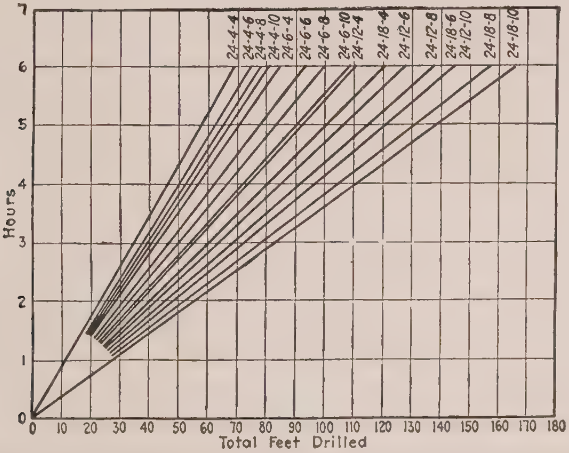

FIG. 49.—Estimated footage drilled with plugger —assuming 24-in. steel change, holes 4, 6, 8, and 10 ft. in depth, and drilling rates of 4, 6, 12, and 18 in. per min.

Spacing, Depth of Holes, Drill Rounds.—Drill holes are placed in groups for specific purposes, such as advancing the face of a drift, crosscut, raise, or tunnel, sinking a shaft, or stoping. The group of holes is called a round. The depth of the round depends upon the depth and angle of the drill holes. The number of drill holes in a round depends upon the area of the face and the type of ground; large areas call for more holes, although the smaller the area the greater the relative number of holes, as smaller areas are tighter. Hard, tough rock requires more holes or a smaller area per hole than do softer rocks. Faces that are not bound on all sides, as is a shaft bottom or a drift end, require fewer holes. Stopes often present two or three faces and require fewer holes and less explosive for breaking. Benches in open pits present two free faces; holes are drilled either vertically at intervals from the top of the bench or inclined or horizontally at intervals along the toe of the bench. High benches may be drilled by a line of holes on the top and also by a line of toe holes. The burden of rock that each hole is expected to break involves both the area enclosing the drill hole and the depth of the hole; the deeper the hole, the greater its burden. The weight of explosive in each drill hole is also determined by the burden as well as by the strength of the rock.

The pattern of the round is laid out with reference to the position of the drill column, the drilling positions feasible with a minimum shifting of the drill, and the cross section. Angle holes are placed so as to unkey the face, others to break to this area, and still others, close to the periphery, to complete the break. Excessive overbreak is to be avoided. Experience is an important factor in determining round patterns. Drilling is the greatest time-consuming operation; the fewer the drill holes, the fewer the drill shifts; the more rapid the drilling, the less time is required. Hard-hitting, faster drills, sharp steel, and skilled drill runners increase drilling rates. Power-feed drifters reduce the time for drill changes and are an important factor in high-speed tunnel driving. Jumbos reduce the time for setup and for the removal of equipment when drilling is finished.

In many mines 10 ft. is the maximum length of drill steel used, and this hole depth is used principally in stoping. There is a tendency to use longer holes, and detachable bits have stimulated changes in practice in the direction of greater hole depth, fewer setups, and greater burdens per hole in development and stoping operations. The results obtained in long-hole drilling in sublevel stoping have likewise been a factor in encouraging departures from conventional practice. In small mines, however, practice is conservative, and often the inability to obtain capital for newer equipment has resulted in the retention of many old-time methods and equipment. Operators of large ore bodies and large mines find it economical to try out new methods and new equipment, as the possibilities of making even small improvements in mining practice often result in large money savings. In the smaller properties such savings may not be sufficient to justify the capital expenditures.

Detaching Detachables.—Screw-thread detachables are removed from drill rods by striking one of the wings of the bit with a hammer in a direction to loosen the bit. Threads are left-hand, as the direction of rotation of the drill steel is clockwise. Socket-type detachables are removed by a U-shaped wedge and reaction block or by a heavy weight slipped over the shank end and dropped, the shoulders of the hole in the block striking the wings of the bit. Special tools are supplied to the miner for different designs of detachables. Their use presents some difficulties, as when lost they sometimes turn up in the ore stream, where they may damage rock breakers and other milling equipment unless removed by magnets or other means. Mounting the tool upon a base of sufficient size may prevent its loss.

Stuck and Broken Steel.—The removal of stuck steel that defies ordinary expedients, such as hammer-and-drill operation, calls for much ingenuity on the part of the drill runner. An open collar with internal diameter slightly larger than the drill steel acts as a clamp when canted and affords an opportunity to use a screw jack, sledge, or hand-held drill. W. A. Knoll and J. C. Sullivan describe drill-rod pullers for collared steel in down holes. These consist of a rectangular steel frame with a bottom yoke to slip under the collared portion of the stuck steel and a top head that carries a tongue, which in turn fits the chuck of an inverted hand-held drill used to loosen the steel. Another design permits the use of two inverted hand-held drills. Broken steel within a hole can be fished for by means of a ring small enough to enter the hole and encircle the steel and thus to act as a clamp when canted. It is manipulated by two cords or wires. Another device consists of an overcast or socket on the end of a rod that can be driven down on the steel in the expectation that its inside corrugations will secure a sufficient hold to enable the steel to be withdrawn. Small broken bits or detachables can sometimes be removed by a drill rod with a long tapered point or by a split fork on the end of a rod and enclosed within a pipe that can be pushed down upon the fork, closing its jaws and holding the fragment (Knoll and Sullivan). A hooked rod is sometimes used for the larger diameter detachables. A simple type of pipe overshot may also be useful. All such tools are designed to cope with individual conditions. When broken or stuck drill steel defies all efforts, a new hole must be drilled.

Drill-sharpening Equipment.—Special shop equipment has been designed for reconditioning and even manufacturing detachables at mines. Grinders driven by electric motor, gasoline engine, or compressed-air engine are common shop equipment and are used for sharpening and gage preparation. Hot millers are also widely used. The drill sharpener is used for sharpening conventional steel, for shank forging, and for rod preparation. Specially designed oil-fired furnaces and quenching troughs with controlled temperatures are available for hardening. Hand-operated threaders are provided for small shops.

Grinding Detachables.—Worn detachables may be sharpened and brought down to the next smaller gage by the use of a grinding machine. This consists of two grinding wheels upon opposite ends of a motor shaft, one wheel being used for sharpening the bit and the other for bringing it to gage size. Both wheels are equipped with accessories for holding and manipulating the bit. A coolant is necessary, as pressure (by hand or foot lever) is used and overheating is possible. Collets used for holding the bit are designed to permit rapid changing. The gaging head is adjustable for different clearance angles. The grinding machine best serves the smaller operator who must recondition his own bits. Its use restricts the regrinds to three or four except where reaming wear is small.

Hot Milling.—Sharpening of conventional steel and of detachables has been greatly speeded up and improved by the use of hot millers. The milling cutter is from 6 to 10 in. in diameter with the peripheral faces meeting at such an angle as to form the cutting surfaces of neighboring wings, the bit being turned 90 deg. for each cut. The sides of the cutter are also toothed for gaging the bit. Bits are heated to a forging temperature before re-dressing and are held in a collet while being milled and gaged. A reaming attachment is used for clearing the hole in the bit. Milling cutters are also designed for six-point and for Carr bits.

The hot miller is a well-designed shop tool, either direct driven by an electric motor or by a motor and V-belt drive. The larger machine requires a 5-hp. motor with cutter speed of 3,500 r.p.m.; the smaller machine uses a 3-hp. motor. The large machine handles from 110 to 180 detachables per hour; the smaller, 75 to 80. Hot-milled bits are allowed to cool and are then heated to a hardening temperature in a special furnace and quenched in cooling troughs of the tray type, in which the cutting edges are submerged at a sufficient depth to produce a rather deep hardened zone. Water is circulated in the cooling trough so as to maintain a uniform water temperature. Hardening temperatures are controlled by a pyrometer or may be tested by a special magnet.

Hot-milling equipment and accessories are designed for large operations where a considerable number of drills and detachables have to be prepared. Even in the smaller shop with the older tool-sharpening equipment the hot miller is a useful accessory. With detachables hot milling gives a greater number of reuses than is possible with grinding equipment. Drill sizes and clearance angles of both grinders and hot millers are more accurately maintained, and drilling is improved by sharp, uniformly sized bits.

Churn-drill Bits.—The bit section of the drill string is designed with a threaded shank for attachment to the drill stem. The body of the bit is flattened, with rounded sides and two deep mud grooves. Several designs (see Fig. 40) are in use, including the chisel shape (6-in. diameter or smaller), the single cutting edge with cupped face, and the Z bit. The cupped face with single penetrating edge or type G bit (Bucyrus-Erie) is widely used in hard-rock drilling; the Z bit is preferred by some for seamy and fissured ground but is more difficult to forge. A 9-in. bit is 5 ft. in length and weighs 500 lb. It is re-dressed and hardened until it is about half its former length by the time it is discarded.

A power-driven bit dresser of special design is used in preference to hand dressing, which usually results in poorly formed bits and requires a number of reheatings. The bit dresser includes an upsetting die which restores the diameter and forms the penetrating edge and cupped face of the bit; a radial hammer forms the reaming surface and brings the bit to gage; the reaming face is parallel to the bit shaft; and the clearance angle is 8 deg. Two or three heats are required for dressing and one for hardening. Two men are required, sometimes three. Furnaces for heating are of special design and are oil-fired. After forging, bits are allowed to cool and are then reheated to a hardening temperature and plunged vertically in salt water to a depth of 1½ in. Cooling troughs for this purpose are of special design. A magnet can be used for determining hardening temperature. At New Cornelia a bit is resharpened 63 times and at Morenci 40 times; at the former 12 bits per shift and at the latter 22 bits per shift are handled. Cost at New Cornelia was $2.12 per bit, including steel loss (E. D. Gardner and M. Mosier).

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CHAPTER V

ROCK BREAKING

The excavation of rock involves breaking, loading, and transportation. Breaking is in most cases accomplished by the use of explosives confined in drillholes or boreholes or in chambers excavated within the mass of rock to be broken and reached by small tunnels or shafts. It may also be done by undercutting and allowing the weight of the mass to cause caving. Further breaking is caused by the movement of the fallen mass in reaching equilibrium.

The physical characteristics of a rock mass that enter into the breaking problem are the hardness, toughness, brittleness, softness, or plasticity of the rock itself and the presence of bedding planes, sheeting planes, joints, cleat, or rift in the rock mass. A rock may be both hard and tough or hard and brittle, brittle and soft, soft and plastic, or soft and friable. Soft rocks are easily drilled and broken, while hard, tough rocks are difficult to drill and require larger amounts and different kinds of explosives. The material encountered in mining ranges from noncoherent, such as earth and sand or gravel, to the most strongly coherent, as granite, diorite, or diabase. Noncoherent and very soft rocks can be loosened by means of the pick, plow, steam shovel, or mechanical excavator, while moderately coherent rocks are preferably broken and loosened by blasting. The wide range in the physical nature of rocks and ores makes it difficult to compare them in any precise manner. The relative hardness and toughness of the more common rocks, together with the energy per square foot of fracture surface required for fracture, are given in the accompanying table. The figures apply to a special group of rocks and cannot be taken to apply to all rocks of the kinds named.

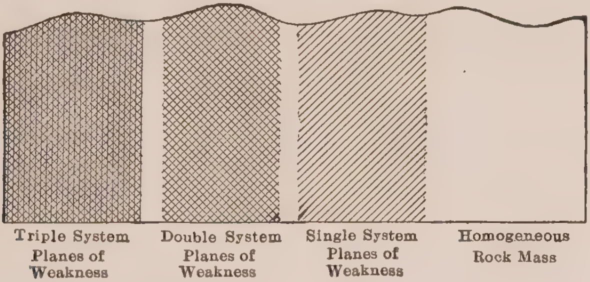

Triple System Planes of Weakness
Double System Planes of Weakness
Single System Planes of Weakness
Homogeneous Rock Mass

FIG. 50.—Weakening of rock masses by fracture planes.

In making use of the table it should not be overlooked that the weakening of a rock mass by the presence of shear planes, or incipient weakening of whatever nature, has an important influence in the selection and amount of explosive required. Thus rock masses may range from solid rock through all the stages to a rock completely shattered and of a mosaic-like structure (Fig. 50). It is evident that solid rock masses require the maximum amount of explosive. If the rock is hard and tough an explosive of high power will be required. If the rock mass is very completely shattered, the explosive serves to unkey the parts of the mass and black powder may suffice for this purpose.

TABLE 16.—PHYSICAL PROPERTIES OF ROCKS[a]

Rock	Relative toughness	Relative hardness	Ft.-lb. per square foot of fracture	Sp. gr.
Fresh diabase	3.0	2.19	624.9	2.95
Altered diabase	2.4	2.11	499.9	
Fresh basalt	2.3	2.05	479.1	2.90
Hornblende schist	2.1	2.00	437.4	3.00
Diorite	2.1	2.17	437.4	2.90
Hornblende granite	2.1	2.20	437.4	2.76
Rhyolite	2.0	2.14	416.6	2.60
Quartzite	1.9	2.20	395.7	2.67
Biotite gneiss	1.9	2.03	395.7	2.76
Augite diorite	1.9	2.13	395.7	2.98
Altered basalt	1.7	1.87	354.1	
Gabbro	1.6	2.15	333.2	3.00
Chert	1.5	2.33	312.4	2.50
Calcareous sandstone	1.5	1.00	312.4	2.66
Granite	1.5	2.17	312.4	2.65
Slate	1.2	1.38	249.9	2.70
Andesite	1.1	1.65	229.1	2.50
Limestone	1.0	1.53	208.3	2.70
Mica schist	1.0	2.08	208.3	2.80
Amphibolite	1.0	2.27	208.3	3.00
Dolomite	1.0	1.77	208.3	2.70
Biotite granite	1.0	2.03	208.3	2.64
Hornblende gneiss	1.0	2.05	208.3	3.02

[a] Compiled from Public Roads, *U.S. Dept. Agr. Bull.* 31.

Compiled by W. O. Snelling and presented in a paper before the Engineering Society of Western Pennsylvania.

EXPLOSIVES USED IN MINING

Many commercial brands of explosives are in use. They are graded in different ways, either by weight or by volume on a percentage basis as compared with a nitroglycerin dynamite or blasting gelatin. "Dynamite" is a general term applied to high explosives in contrast to black powder, which might be termed a low explosive. "Permissible explosive" is a term applied to a group of explosives used under coal-mining conditions and approved for such use by the Bureau of Mines.

Explosives when fired produce a flame. That of black powder is the longest and greatest in duration. Dynamite gives a shorter and hotter flame, and permissibles have a smaller flame of brief duration and lower temperature, to avoid ignition of gas or dust. Explosives produce gases or fumes such as carbon dioxide, nitrogen and steam, which are nontoxic, and carbon monoxide, nitrogen oxides, and hydrogen sulphide, which are toxic. Some carbon monoxide is always present; presence of nitrogen oxides may result from improper use of explosives, and hydrogen sulphide is sometimes present. Toxic gases are termed "fume," and explosives are classified, according to amounts of toxic gases liberated, as very good (for the smallest amount), good, fair, and poor. As indicated in Table 17, explosives liberate relatively large amounts of heat, and high temperatures may be attained. At such temperatures the liberated gases exert great pressures when confined within a blasthole.

TABLE 17.—QUANTITY OF HEAT AND ESTIMATED TEMPERATURES
(*From I. Grageroff, Am. Min. Cong. Jour.*)

	1 lb. nitro-glycerin	1 lb. blasting gelatin	1 lb. 40 % gelatin	1 lb. 35 % gelatin	1 lb. black powder
B.t.u.	2850	2960	1800	1400	1200
Temperature, deg. F	6280	6400	6025	5470	5020
Cu. ft. gas at above temperature	156.7	....	....	....	51

Sensitiveness to detonation, shock, friction, and handling is an important characteristic, some explosives requiring careful handling and others being relatively insensitive. Rate of detonation, measured in feet per second, determines the blasting effect of an explosive. Physical state, expressed as rubbery, plastic, cohesive, or granular, affects its use. Inflammability is another factor. Density, conveniently measured by the number of 1¼- by 8-in. cartridges per 50-lb. box, is important. Resistance to water must also be taken into consideration.

All explosives are potentially dangerous and require careful handling and storage by experienced men. Fortunately the manufacture of explosives is in the hands of relatively few companies, and these have standardized their products, improved the quality, and publicized precautions and safety measures. Explosives are manufactured and packaged to comply with specialized use in metal, coal, and other mines and in open pits and quarries. Table 18 indicates quantities and kinds of explosives used in the United States in mining and construction.

TABLE 18.—CONSUMPTION OF EXPLOSIVES IN 1940, UNITED STATES
(U.S. Bureau of Mines)

	Black blasting powder, 25-lb. kegs or cases			High explosives, lb.	Permissible explosives, lb.
	Granular	Pellet	Total		
Coal mining	937,865	1,193,909	2,131,774	21,688,322	57,820,655
Metal mining	9,424	184	9,608	106,000,338	28,750
Quarry and industrial mineral mining	94,123	18,009	112,132	59,634,470	452,800
Railway and construction	93,857	8,058	101,915	102,059,140	75,330
Other	28,072	6,636	34,707	10,799,430	58,595
Total 1940	1,163,341	1,226,795	2,390,136	305,179,700	58,436,130
1941	1,115,280	1,263,023	2,378,303	351,957,453	70,612,209
1942	966,665	1,254,685	2,221,350	359,698,547	84,022,296

Chemical Composition.—Nitroglycerin, nitrocotton dissolved in nitroglycerin, and ammonium nitrate are the important explosive compounds. Alfred Nobel invented dynamite in 1867 by using diatomite as an absorbent for nitroglycerin. Diatomite is inert and is no longer used for this purpose. Absorbents are of carbonaceous materials, such as wood meal, flour, and starch, and mixtures are so made as to provide a sufficiency of oxidizing constituents to oxidize them. Freezing-point depressants (nitrotoluenes, nitrated sugars, tetranitrodiglycerin, and nitroglycol) are incorporated in the mixtures to a sufficient extent to avoid freezing of explosives. (The term "low-freezing" has been largely discarded.) Straight dynamites are mixtures of nitroglycerin, carbonaceous absorbents, depressants, and other materials; gelatins are mixtures of nitroglycerin-nitrocotton and other constituents; ammonia dynamites are mixtures of ammonium nitrate, nitroglycerin, absorbents, depressants, and other materials; ammonia gelatins are mixtures of nitroglycerin-nitrocotton, ammonium nitrate, and other constituents similar to those mentioned; permissibles of the ammonia type contain 10 to 15 per cent nitroglycerin, 60 to 80 per cent ammonium nitrate, a low content of absorbent, and small quantities of other ingredients; coarse ammonium nitrate permissibles contain coarse granules of ammonium nitrate, a low content of nitroglycerin, and low-density combustibles.

Black powder is an intimate mixture of sulphur, charcoal, and sodium nitrate in proportions to cause complete oxidation. It is broken into granules and sold in seven standard sizes ranging from CCC, the coarsest and slowest, to FFFF, the smallest and fastest burning. Granules are glazed with graphite to improve the free-running qualities. Another form is pellet powder. The black powder is pressed into cylinders 2 in.

in length and 1¼ to 2 in. in diameter, each pellet having a ⅜-in. diameter axial hole for ignition. Four pellets are made up into an 8-in. cartridge, giving an explosive more resistant to moisture and safer and easier to handle than four 2-in. cartridges.

High detonating rate gives an explosive of shattering or smashing power, in contrast to black powder, which has a pushing or heaving effect and is slower in action. Black powder is extensively used in coal mining, as it produces a maximum proportion of lump coal. Its sustained flame, however, makes it unsuitable for gaseous and dusty mines. Dynamites vary in their detonating rates according to different grades. Low-density ammonia dynamites have a rate as low as 4,000 ft. per sec. and highest grade gelatin dynamite 20,000 ft. per sec. Velocity of explosion of black powder is about 1,000 ft. per sec. Foot-pounds of energy per pound of weight as given by E. H. Rockwell are 2,204,000 for nitroglycerin, 1,500,000 for guncotton, and 1,030,000 for black powder.

Water resistance is highest for the gelatins and gelatin-ammonia dynamites and least for granular black powder, which is unsuitable for wet conditions. Straight dynamites, ammonia dynamites, many permissibles, low powders, and bag powders have only a limited water resistance. "Water resistance" is a relative term. Wet conditions range from a damp drill hole to one in which water stands. Cartridge wrappings offer some resistance to water; they vary from sprayed to sprayed and dipped in paraffin; some are waxed and some have the ends closed by wire loops; other cartridge designs are water-resistant; nitromon is packed in sealed metal cans. Care is required in the selection of explosives to suit the various wet conditions.

"Sensitiveness of an explosive" is used with two significations, one relative to its detonation and the other with respect to handling. High, straight dynamites are probably the most sensitive with respect to handling and for this reason have been largely supplanted by safer dynamites. Dynamites now in use can be detonated by No. 6 or No. 8 blasting caps or by detonating fuses. Certain quarry explosives such as nitromom are so insensitive as to require special primers. The primer cartridge for use with electric blasting caps contains 6 in. of amatol (TNT and ammonium nitrate) at the top of the can, the remainder being occupied by nitromon. Caps are placed in tubes extending down into the amatol and the primer is used at the top of the charge. Another design has the amatol in the center portion of the can, along the outside of which is a fluted metal tube or tunnel through which the Primacord (detonating fuse) is run. Center-type primers can be placed at bottom, center, or top of charges.

"Sensitiveness to propagation" is used in reference to an explosive that can be exploded by another near-by charge not in contact with the charge. Straight dynamite is sometimes used in wet soil for ditching. A line of holes at 18-in. intervals is loaded with explosive, the first hole of the series being detonated by an electric blasting cap and the others by propagation. Detonation is propagated in charges of explosives where individual cartridges are in contact.

Production of toxic fume depends upon the kind of explosive and how it is used. Straight dynamites, 40 to 60 per cent, are not suitable for underground use. The gelatins in the lower grades, 20 to 60 per cent, are rated excellent. The higher grades, 75 to 100 per cent, are rated very poor to poor; ventilation must be adequate when they are used underground. Ammonia dynamites are rated fair; their use underground requires adequate ventilation. Permissibles are rated fair to good and are used underground where ventilation is ample. Black powder gives bad fumes, and ventilation must be adequate where it is used underground.

Pour-type explosives are so manufactured that they can be poured through funnels into vertical drill holes or can be handled by loading machines. They are granular and free running. Plastic explosives have the advantage that they can be expanded to fill the drill hole and remain in place in uppers. Cartridges are sometimes slit on the sides to facilitate expanding of the cartridge to fill the drill hole. Pellet powders are unyielding.

Containers.—Explosives are made up into cylindrical paper cartridges; granular and pour types are contained in bags; black blasting powder is in 25-lb. metal kegs. Cartridges are packed in 50-lb. wooden boxes, marked with cartridge size, grade of explosive, and manufacturer's name. Bags are packed four to a 50-lb. box.

Cartridge sizes vary, the commonest being 1¼ by 8 in. Smaller sizes are obtainable; they are often made to conform to user's requirements. Pellet powder is made up in 1¼-, 1⅜-, 1½-, 1¾-, and 2-in. by 8-in. cartridges. Quarry powders for churn-drill holes are made up in sizes 4½, 5, and 5½ in. in diameter by 16 in.; 5 and 5½ in. by 25 in.; 8 by 18 in., and 8 by 27 in. (50 lb.). Nitromon is packed in can sizes 4 to 7½ in. in diameter by 24 in., 8 by 21 in., and 9 by 19 in. Tamptite cartridges are of special design to obviate slitting of cartridges. Another type consists of a perforated paper shell, the perforations following a spiral line. Tamping causes the pressure to divide the paper shell, allowing the explosive mixture to expand and fill the bore. Spiralock consists of an outer heavy paper sleeve with an inner thread to accommodate a specially wrapped cartridge with a matching spiral thread. Sticks of explosives used in seismograph work can be readily made up and more conveniently used than ordinary cartridges.

Cartridge count, based upon a 50-lb. box, is a convenient way to determine the density factor of an explosive. Explosives of high density, greatest energy or weight in smallest volume, give a low count, 85 to 100 cartridges (1¼ by 8 in.) per 50-lb. box. Ammonia dynamites range from 128 to 189, and high cartridge counts reach 256 per box. Relative explosive energy can be figured by the percentage strength of the commercial explosive and the weight of the charge. Some commercial explosives are classified in Table 19.

TABLE 19.—CLASSIFICATION OF COMMERCIAL EXPLOSIVES
(*Atlas*)

Type	Per cent strength weight	1¼″ × 8″ cartridge count; 50-lb. box	Rate of detonation, ft. per sec.	Water resistance	Fumes
Blasting gelatin R	100	100–110	20,000	Excellent	Fair
Gelatin VP	20–100	85–110	6,500–20,000	Excellent	Very good-poor
Ammonia gelatin P	30–90	85–110	7,000–20,000	Very good	Excellent-poor
Semigelatin FP	65	107–126	10,000–11,000	Good	Very good
Nitroglycerin dynamite VC	20–60	105–115	8,800–19,000	Fair	Good-poor
Ammonia dynamite C	15–60	105–115	7,300–9,000	Limited	Fair
Ammonia, low density Dynamite:					
C	65	128–189	7,000–9,000	Limited	Fair
G	20–70	Large cartridges	6,500–19,000	Limited-very good	Poor
C	65	133–170	8,000–10,000	Limited	Fair
G	20–65	Bags	4,000–5,500	Limited	Very good
G	10–65	Bags	4,000–5,500	Limited	Very good
Permissibles:					
Gelatin P	35–55	97–126	8,200–16,400	Very good	Good-fair
Dynamite C	55–60	133–256	4,500–10,000	Limited	Fair
Low powder G-C	5–20	100–110	4,000–9,500	Limited	Good-poor
Pellet powder	......	106–125	....	Limited	Poor
Black blasting	......	25-lb. keg	....	Poor	Poor

NOTE: Consistency is denoted by letters as R—for rubbery; VP—very plastic; P—plastic; FP—fairly plastic; VC—very cohesive; C—cohesive; G—granular.

Liquid Oxygen Explosive.—Lox, or liquid oxygen explosive, for a time attracted wide attention and was used sufficiently to expose its limitations. Lox consists of a canvas cartridge filled with granular carbonaceous material, moistened, which when soaked in liquid oxygen, removed from the soaking box, and placed in the drill hole can be detonated and is an effective blasting agent. A primer consisting of a dynamite cartridge containing an electric blasting cap is necessary for detonation. The cartridge loses its oxygen by evaporation and consequently must be soaked and used with a minimum loss of time. Underground use of lox presents too many difficulties; its special field appears to be in open-pit operations where large-diameter cartridges can be used in churn-drill holes. Chuquicamata made use of lox in its open pit from 1926 until 1929 when, as a result of a bad accident, its use was discontinued. Present use of lox in the United States appears to be restricted largely to some coal strip pits. References in the bibliography at the end of Chap. VI will give other details.

Nonexplosive Agents.—Cardox employs liquid carbon dioxide enclosed in a steel tube within which is a heating element that is ignited by an electric squib. The tube contains a shearing disk at the lower end of the tube and ports for the release of the gas within the drill hole. Discharge pressures can be adjusted from 10,000 to 20,000 lb. per sq. in. Tubes weigh from 10 to 40 lb. and can be recharged and reused many times. They are placed in the drill hole and tamped with clay stemming; an electric current of 2 or 3 volts and a fraction of an ampere is required for ignition. Cardox is effective in producing lump coal and is used in coal mining. It produces neither fume or flame. Storage units for the liquid carbon dioxide and charging units have been designed for this special application.

An interesting development of the hydraulic cartridge was sponsored by du Pont, but war conditions temporarily took it off the market. The rig consists of a motor-driven pumping unit mounted upon skids or car and delivering high-pressure oil through a high-pressure hose to an expansible mining tube inserted in a drill hole. The tube or pressure stocking is inflated and exerts sufficient pressure upon the coal to break it down. Its use avoids flame, gas, and dust. Pressure pads for use in top cuts represent another development in coal mining.

Firing Explosives.—Squibs and fuse are used to ignite black powder charges. A fuse with attached blasting caps is used to detonate dynamite charges and explosives that respond to detonation. Electric blasting caps are used with detonating explosives; the electric squib is a special form used to ignite black and pellet powders. A fuse requiring an electric blasting cap for its detonation is used with detonating explosives. In all charges of explosives it is necessary to bury the end of the fuse within some part of the black powder charge, to bury the blasting cap or electric blasting cap within the end of one cartridge, or to lay the detonating fuse alongside of and in contact with the charge.

One or more charges up to a limited number may be fired by fuses or fuses with attached blasting caps; but as each individual fuse end must be ignited, it is impracticable to handle a large number without special devices. This method is restricted to small headings, stopes, chute blasting, and bulldozing. Electrical blasting is almost universally used for shafts, large headings, bench blasting, major blasts, and groups of holes. Detonating fuse is more or less restricted to quarrying and open-pit operations, but it may also be used in longhole blasting in pillar and stope mining. In all such blasting, electric blasting caps are necessary as accessories to the detonating fuse.

Fuse.—A safety fuse consists of a powder train enclosed within three or more protecting and waterproofing layers; powder trains consist of a free core or semisolid core; commercial grades are distinguished chiefly by varying degrees of water resistance. Ignition speeds are approximately 90 to 120 sec. per yard when burned in the open at sea level. A fuse is sensitive to dampness and dryness and is brittle at low temperatures. Pressure caused by tamping may appreciably increase its ignition speed, while high altitudes may retard it. Fuse is supplied in small packages containing two 50-ft. coils and on reels containing 3,000 ft.

Detonating Fuse.—A lead tube filled with trinitrotuolune (Cordeau-Brickford) was used in open-pit operations but has been displaced by Primacord, which contains an explosive core of pentaerythritetetranitrate (PETN) within a waterproofed textile covering. Its insensitiveness, light weight, and high detonating rate (20,350 ft. per sec.) make it effective in detonating column charges and deck loads. Three types are available: Plain, reinforced (for deep holes), and wire-bound (for ragged deep holes), in spools of 500 and 1,000 ft. Weights of the 500-ft. spool for the three types are, respectively, 11, 12, and 19 lb. It is weather- and heat-resistant and can be stored without deterioration. It is water-resistant except in extremely wet conditions.

Blasting Caps.—High explosives, the dynamites and permissibles, require the use of a detonator or blasting cap for their explosion. The conventional blasting cap (Fig. 51) consists of a cylindrical shell, usually copper, of 0.22 in. internal diameter and various lengths, charged at the closed end with a compound consisting of a mixture of fulminate and potassium chlorate, trinitrotoluene, lead azide, nitrormannite, or other detonating mixture. Priming and ignition charges are placed upon the detonating mixture, leaving a ¾-in. space for the insertion of the fuse. The end of the safety fuse is cut square and inserted into the cap in contact with the cap charge, and the shell is crimped about the fuse to form a tight joint. Hand and bench crimpers of good design are available. An oilless waterproofing compound is often placed at the joint as an additional precaution against the entrance of moisture. Varying the lengths of safety fuse enables blasts to be exploded in sequence. A difference of 2 in. in length of respective groups of blasts is sufficient for sequence. No. 6 and No. 8 caps are in general use.

Electric Blasting Caps.—Electric blasting caps are of two kinds, instantaneous and delayed-action. Both rely upon the heating to incandescence of a platinum bridge between two insulated lead wires upon the passage of a sufficient current; the hot bridge wire ignites an ignition mixture, which in turn ignites the cap charge in the instantaneous type and the timing element in the delayed-action type; the timing element in turn ignites the detonating mixture. The timing period between delays is about 1½ sec.; delays are supplied in a series of 10 from no delay up. Several electric blasting cap designs are shown in Fig. 51.

Electric blasting caps are sufficiently waterproof for ordinary blasting conditions. Caps of special construction with high water resistance are manufactured for submarine blasting or for severe water conditions.

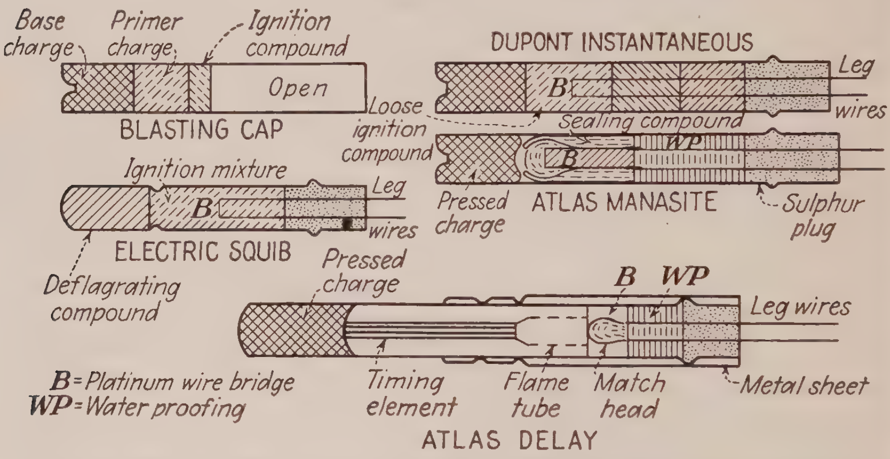


FIG. 51.—Details of construction of blasting caps, electric blasting caps, delays, and electric squib.

Electric squibs (Fig. 51) are similar in construction but contain a deflagrating mixture, which when ignited bursts the shell and imparts a hot flame. They are used to ignite black-powder and pellet-powder cartridges and have displaced the miner's squib.

Firing Electrically.—Where electric detonators are used in blasting charges, a group of charges may be joined in series to a pair of lead wires connecting with the source of current. One leg wire of No. 1 cap is connected to a leg wire of No. 2 cap, the second leg wire of No. 2 cap to the first leg wire of No. 3 cap, and so on. The free wire of No. 1 cap and the free wire of the last cap in the series are connected to the two lead wires. The series hookup is advantageous in that such a circuit may be readily tested with the circuit tester. It is generally used with blasting machines. Connecting the leg wires from each cap, one to each lead wire, gives a parallel circuit. For a given number of caps the resistance

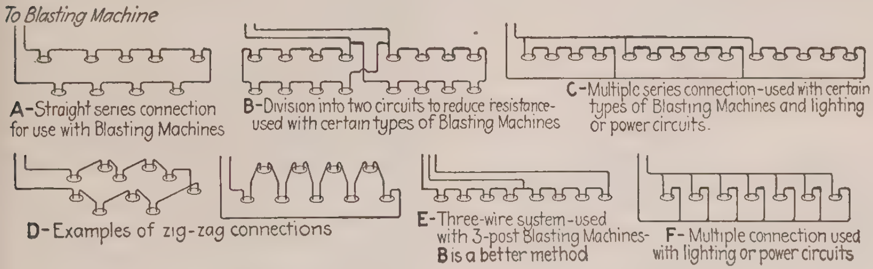
TABLE 20.—RESISTANCE OF ELECTRIC BLASTING CAPS
(*du Pont*)

Leg wires, ft.	Resistance, ohms per cap				
	Regular and waterproof	Delays	Leg wires, ft.	Regular and waterproof	Delays
4	1.26	1.04	12	1.52	1.30
6	1.32	1.10	20	1.79	1.56
8	1.38	1.17	30	1.76	1.54
10	1.45	1.23	50	2.18	1.96

NOTE.—For copper leg wires, resistance for the connecting wire is 10.15, 12.8, and 16.14 ohms per 1,000 ft. for respective Brown & Sharpe gages 20, 21, and 22; for common sizes of leading wires, resistance is 2.53, 4.02, and 6.39 ohms per 1,000 ft. for gages 14, 16, and 18, respectively.

of a parallel hookup is very much less than for a series circuit, the resistance of which is the sum of the resistances of each cap and its leg wires and the exterior wires of the circuit. Another hookup is the combination of series and parallel, groups of caps in series being connected in parallel to the lead wires. Figure 52 indicates several wiring plans.

The resistance of each blasting circuit should be estimated so as to avoid overloading the blasting machine or other source of current. Table 20 gives the resistances of blasting caps and wires. Each circuit should



A-Straight series connection for use with Blasting Machines

B-Division into two circuits to reduce resistance-used with certain types of Blasting Machines

C-Multiple series connection-used with certain types of Blasting Machines and lighting or power circuits.

D-Examples of zig-zag connections

E-Three-wire system-used with 3-post Blasting Machines- B is a better method

F-Multiple connection used with lighting or power circuits

FIG. 52.—Methods of connecting circuits for electric ignitions.

also be carefully tested before final connection with current source. Firing current is supplied by a blasting machine or from a power circuit. The larger blasting machines are designed to carry some 50 shots in series; smaller machines of several types are on the market. Figure 53 illustrates various machines. The two-post machine of the push-down type is designed to fire a single circuit in series arrangement and is widely used in quarry and other mining operations. Power-circuit current is limited to 110 or 220 volts, d-c or a-c. Such circuits are protected by a fuse, switches, and locked boxes to ensure safety.

Blasting machines are tested by using a special rheostat that enables a resistance equal to the resistance of a given number of caps to be interposed in the circuit of the blasting machine, which is then operated with a blasting cap in series (in a safe position). In other forms of rheostats an indicating light is used in place of the blasting cap. The rheostat is shown in Fig. 54. For testing blasting circuits a special kind of galvanometer is employed. This contains a silver chloride cell that supplies a very small current in series with the galvanometer and the circuit to be tested. The galvanometer-needle deflection indicates the current passing. Blasters' handbooks supplied by manufacturers of

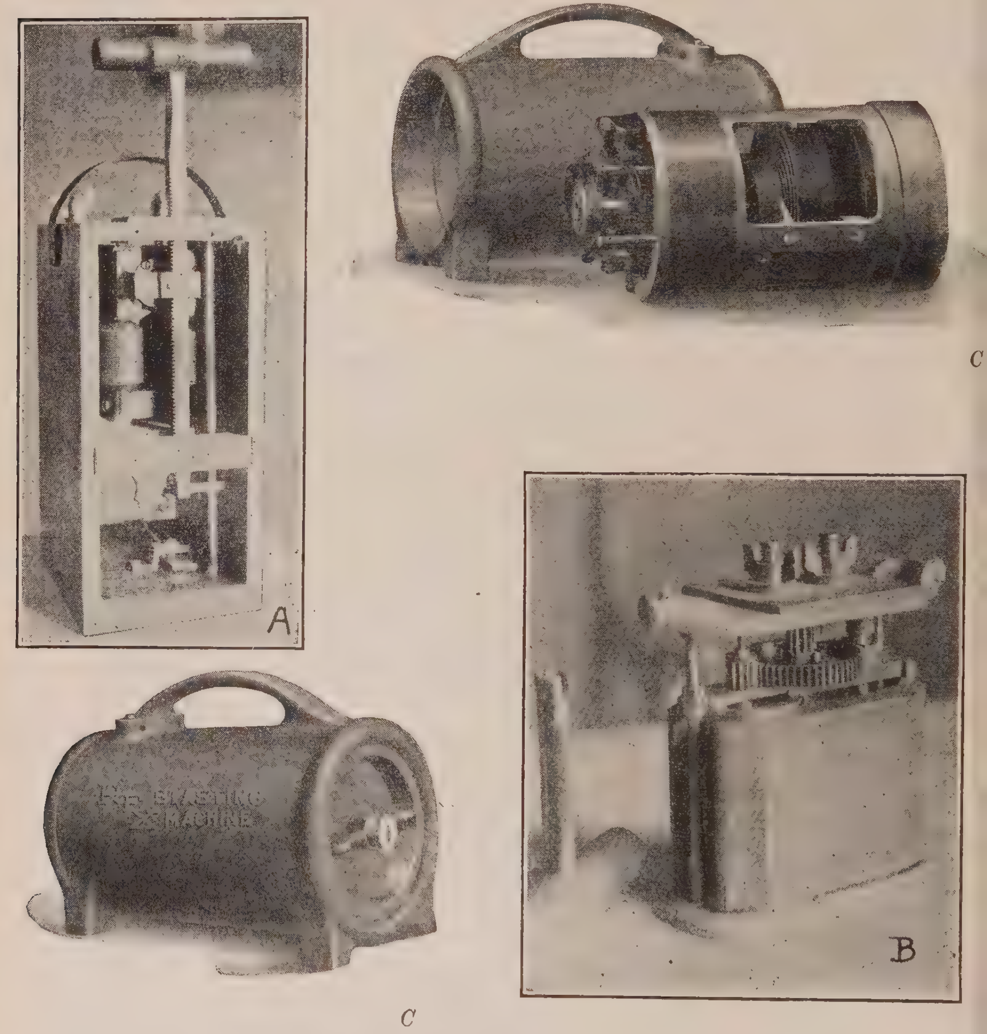


Fig. 53.—Blasting machines. *A* is the ordinary 2-pole machine; *B* is a 5-shot hand machine, operated by twisting the handle (outer case of machine removed); *C* is a high-amperage machine: upper right-hand view shows interior mechanism. (*California Cap Co.*)

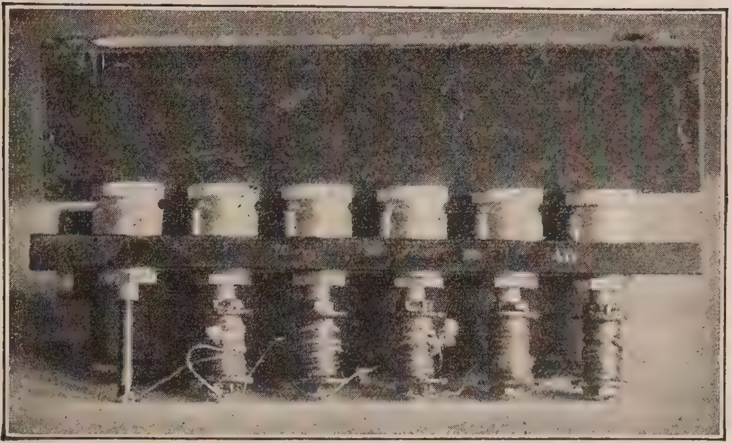


FIG. 54.—Rheostat for testing blasting machines. The numbers indicate the numbers of caps the machine will explode with one cap in series with the rheostat. The resistance plugs have been removed from the wooden case. (*California Cap Co.*)

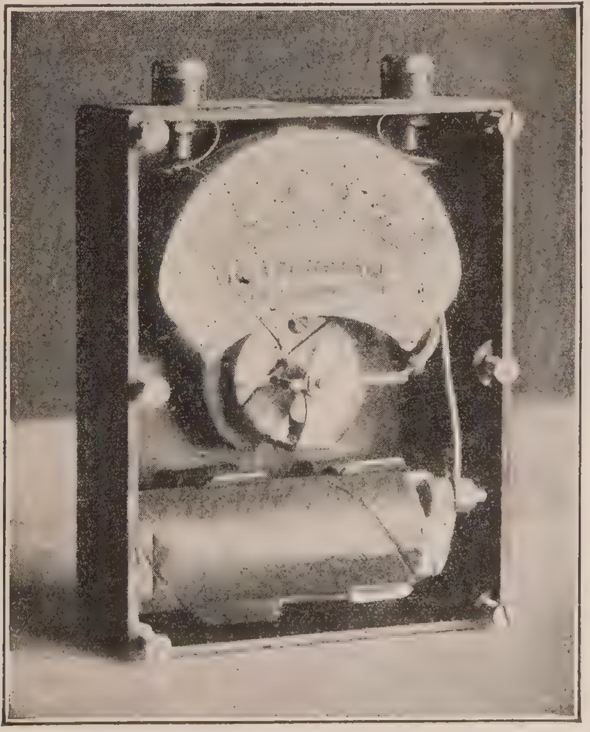


FIG. 55.—Galvanometer for testing blasting circuits. (*California Cap Co.*)

explosives give details for estimating the resistance of blasting circuits and for electrical blasting in general. Figure 55 illustrates the blasters' galvanometer.

In electrical blasting stray currents are a source of danger and might cause premature explosions by entering the blasting circuit. Thunderstorms are potentially dangerous during blasting operations. An ore body that is a good conductor can set up dangerous conditions with respect to electrical blasting. The blasting circuit where a power line is the source of current should be kept from any contact with electrical conductors, pipe, rails, and hoisting ropes. Leg wires, lead wires, and blasting circuit should be short-circuited until the blast is ready to be connected. Maintenance of insulation on leg and lead wires, tapping all connections, and keeping such circuits away from wet ground prevent shorting. Careful testing of all circuits before blasting to determine any grounds is necessary. R. G. Clay and C. F. Seamon state that a switch is not safe for underground use and recommend that the insulation between blasting circuit and power source should be an air gap of at least 18 in. They accomplish this by means of a hot box (Fig. 57) protected by a resistor. Contact with the blasting circuit, which is protected by a safety switch, is made by means of a shooting paddle that connects the supply circuit with the safety switch across the gap.

Handling and Storage of Explosives.—All explosives require scrupulous care in handling. Protection from fire, sparks, or other kinds of ignition is necessary in all cases. Detailed methods for handling explosives and safe methods for their use are given in the bibliography at the end of Chap. VI. Most explosives deteriorate when exposed to moisture, weather, alternating high and low temperatures, or when improperly stored. Well-ventilated magazines maintained at uniform temperatures with protection from dampness are essential. Manufacturers of explosives supply detailed information as to the best conditions for the storage of their products.

Caps and fuse are stored in separate magazines, and provision is made for separate containers for primers and explosives where they are being transported to places for use. State laws prescribe magazine construction, distances from buildings, maximum quantities that can be stored, and many regulations designed for safety. Federal laws regulate the transportation of explosives.

Use of Black and Pellet Powder.—Small charges of black powder are best used in cartridge form. After the drill hole is cleaned the cartridges are inserted and then are pushed to the bottom by a wooden tamping stick. The primer cartridge, with the safety fuse inserted and the end of the cartridge paper securely tied about it, is pushed down upon the charge; loose stemming is added and the remainder of the hole is tamped with additional stemming. Sequence in blasting several holes is accomplished by using different lengths of fuse. Pellet powder cartridges have the fuse end inserted in the central hole. Electric squibs are inserted in the central hole of the pellet primer cartridge, which is placed in the middle of the charge. Delayed-action squibs permit sequence blasting.

Chambered holes are filled by pouring the powder through a brass funnel; horizontal holes are charged with a powder charger. Large charges of black powder are exploded by means of a primer consisting of several sticks of dynamite, one of which contains an electric blasting cap, tied in a compact bundle and placed in a central portion of a charge. After the priming charge is placed, the remainder of the black powder charge is poured into the hole and worked down by a tamping rod until all voids of the chamber are filled.

Use of Dynamites and Permissibles.—Explosives that require the use of detonators are prepared as follows: For the smaller drill holes the primer cartridge is pierced diagonally by a wooden or copper skewer on the side just above the mid-point; the detonator, crimped properly to the fuse, is inserted well within the explosive so as to bring the cap end at the axis of the cartridge. A short length of cord is given a half hitch around the fuse about an inch above the cap; the ends are then brought around the cartridge and tied in a firm knot just below where the fuse leaves the side of the cartridge. The primer cartridge may be placed as the second cartridge in the drill hole or as the last cartridge of the charge; stemming is introduced and lightly tamped, after which additional stemming and firmer tamping complete the charging. See Fig. 56.

Primer cartridges for electric blasting caps are prepared as described above. Another method is to unwrap the end of the cartridge, pierce the explosive, and insert the cap. The leg wires are brought to the side and inserted through a hole made through the middle of the cartridge, then tightened and brought up alongside the cartridge. Still another method is to cut the cartridge longitudinally, lay the cap within, and wrap the cartridge in box-lining paper, tying firmly with cord. It is desirable to avoid kinks, sharp bends, and half hitches in the lead wires. The primer cartridge is placed as the second cartridge from the top of the charge with the cap pointing toward the principal part of the charge; in sequence blasting the primer is placed second from the bottom, with the cap pointing toward the opening of the drill hole.

The maximum detonating effect of the blasting cap is from its closed end in a direction parallel with its length. The safety primer, described by T. Soule, consists of a wooden block (Fig. 57) in which the electric blasting cap is placed in an axial hole with the closed end flush with the end of the block; the reverse end of the block is tapered and a saw cut on a

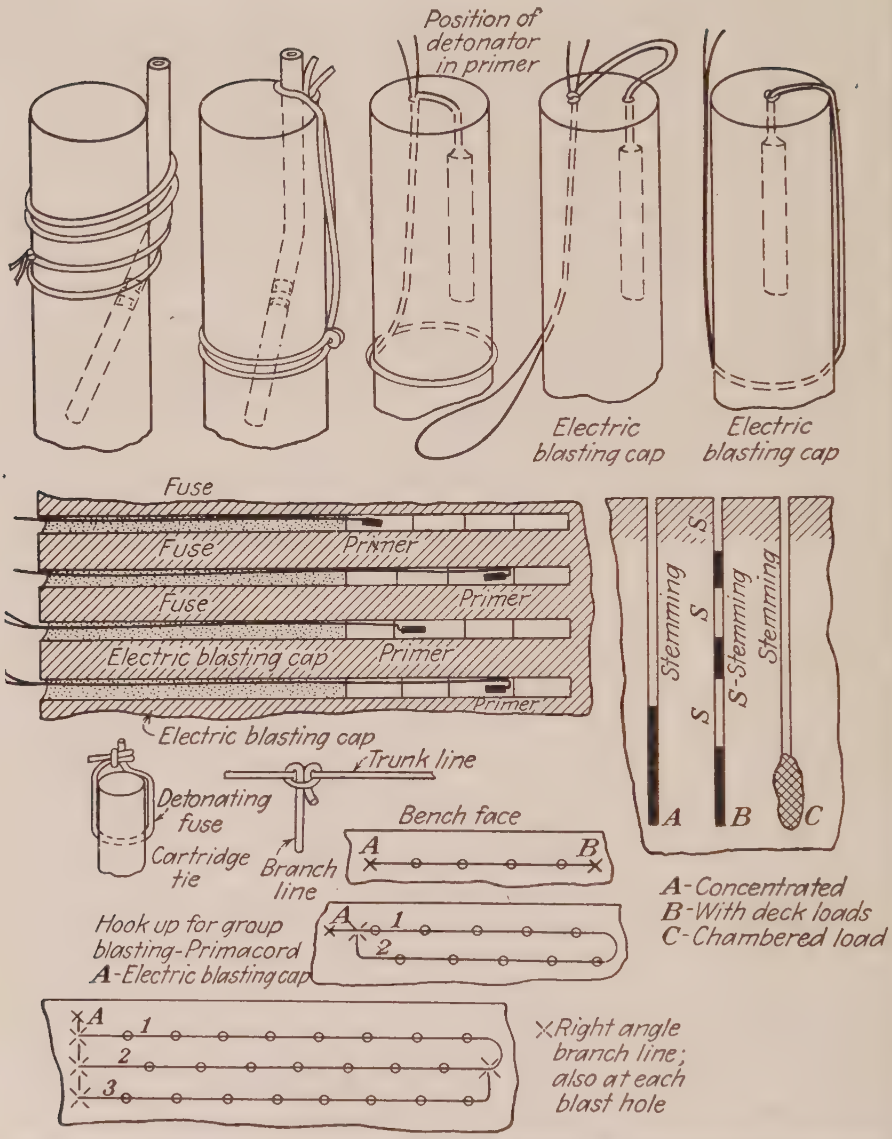


FIG. 56.—Methods of preparing primer cartridge; position of primer cartridge in charge; kinds of loading churn-drill holes; use of detonating fuse.

diameter at this end allows the leg wires to be separated and bent back along the sides of the block. The cap is glued in the hole. The block is about ¾ in. longer than the longest delay cap to be used and is quarter octogon in section so as to allow room for the leg wires to pass. The primer cartridge is prepared by opening and placing the wooden primer with the end of the cap against the explosive; it may also be slit so as to reduce air spaces to a minimum. The primer cartridge is pushed to the bottom of the drill hole and lightly tamped, after which the remainder of the charge is inserted, followed by stemming. In orthodox practice the primer cartridge is never tamped and the following cartridge only lightly tamped. The use of the safety primer permits slitting each cartridge and firmly tamping it to secure concentration of charge.

Churn-drill holes are charged as follows: The first cartridge is pierced about 3 in. from the top. Then Primacord is threaded through, the cut end being brought around the line above the cartridge and firmly tied (Fig. 56). Another method is to pierce an axial hole through the cartridge, thread the Primacord through, and tie a knot in the cut end. The first cartridge is lowered to the bottom of the hole and firmly seated, the Primacord being cut at a point 2 ft. above the collar of the hole and the end being tied to a piece of wood weighted by a rock. Succeeding cartridges may be dropped to the bottom of the hole or they may be cut into several pieces and each piece dropped separately. The best practice is to lower the cartridges one at a time on a rope. At New Cornelia 50-lb. cartridges (8 by 27 in.) are handled by a compressed-air-operated loading machine that consists of a small hoist and a short derrick arm with a sheave mounted upon two rubber-tired wheels and secured in the loading position by means of three pointed steel rods. The

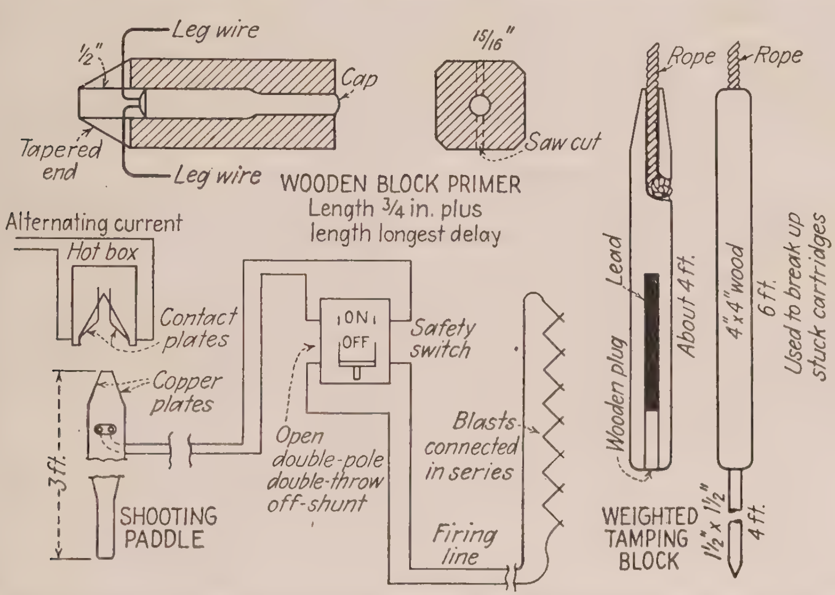


FIG. 57.—Upper left—Soule's safety primer; lower left—safety gap electric switch for electrical blasting; right—tamping block and breaker.

rope is a ⅜-in. rubber hose terminating in a compressed-air-operated release hook. A safety brake is used to control lowering. The use of the large cartridge cuts down air spaces. Stemming consisting of mill tailings is shoveled into the hole. Deck loads are charges placed in the upper parts of drill holes above the main charge and separated from it by stemming. Tamping of charges is not necessary. Where ropes are employed, a closed hook with several sharp edges is used and the cartridge is tied to the hook with cord. Several sharp jerks cut the cord and release the cartridge. Primacord trunk lines are laid along the line of holes and each branch is connected at right angles. A blasting cap with safety fuse is attached by means of a special union at one end of the trunk line; instantaneous electric blasting caps are also used, attached at both ends of the trunk line by means of special unions.

Many serious accidents have occurred in loading churn-drill holes. They may be prevented only by a well-planned procedure and rigid discipline in carrying it out. Where black powder is used, protection from sparks, flame, and impact and grinding friction is essential. Areas about the top of the hole should be cleared, and explosives should be delivered at a distance and brought up to the loading point a small quantity at a time. The Institute of Makers of Explosives recommends gelatin dynamite and low-ammonia explosives for churn-drill holes, never straight dynamite. The use of the tamping block for either compacting explosive or for stemming is to be avoided.

Chambering or Springing.—The loading capacity of small-diameter drill holes can be increased by chambering or springing, which consists of exploding several small charges successively in the hole until a cavity of sufficient size to receive the charge has been made. Soft rocks may require only one springing charge, but hard rocks require up to three. At New Cornelia the hardest rock requires three springing charges in 24-ft. machine-drill holes, 2½ to 1⅞ in. in diameter. The first charge is 5 lb., the second 12 lb., and the third 30 lb. of 60 per cent special gelatin, with stemming respectively 4, 8, and 16 ft. A springing blast develops considerable heat, which is slowly dissipated. Therefore an interval is necessary between springing charges to allow a hole to cool off. No blastholes are sprung twice in the same day, and a blasthole is never loaded until the day after the final springing charge.

Drill-hole Conditions.—Drill holes should be clean. Usually a compressed-air pipe is used to blow them out; scrapers are used where blowing is impracticable. After chambering, cleaning may be necessary and the air pipe is used for this purpose. Churn-drill holes should be examined by flashlight or by reflecting sunlight down by means of a mirror. Caved holes may need to be cleared. Water is bailed out. The condition of the hole, whether wet, damp, or dry, is of importance in selecting the explosive and in determining the interval between loading and firing. At the United Verde open pit, hot holes were tested by thermometer and were cooled by water. The tamping bar is sometimes used in testing a churn-drill hole for clearance.

Tamping.—Tamping sticks are round, of a diameter and length to accommodate the deepest holes. They are made of light wood or of bamboo rods. For churn-drill holes a cylindrical wooden block of hardwood securely attached to a rope and from 3 to 4 ft. in length is employed; it must be weighted where used under water. Tamping blocks are not used in large churn-drill holes. When cartridges stick, a special tamping block (Fig. 57) is used to break them up. This has a pointed wooden prong as an extension. In small-diameter drill holes the tamping stick is used to compact the powder charge and to press the stemming into place.

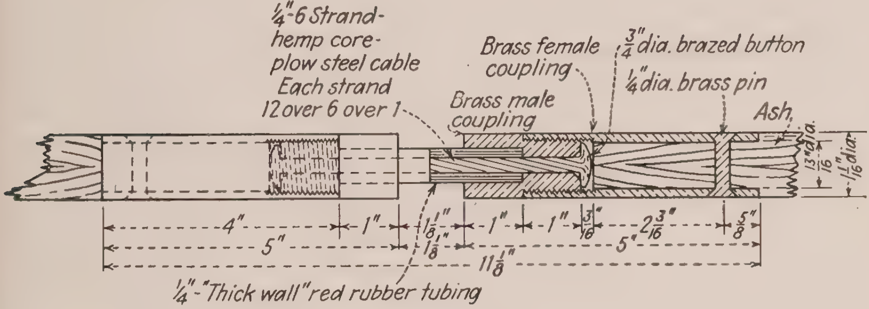


Fig. 57*a*.—Sectional tamping rod for long holes. (*Noranda Mine.*)

Long holes, such as diamond-drill holes, require flexible tamping rods or sectional rods (Fig. 57*a*) with couplings. An air hose plugged at the tamping end is often used. Another design is rod sections joined together by rubber-covered wire cables.

Stemming.—Stemming is frequently omitted where high-power explosives are used and time is important, as in tunneling. In coal mining, stemming is necessary for pellet, black powder, and permissibles. Stemming increases the blasting effect and protects the charge in the interval before firing. A moistened mixture of sand and clay made up into cartridges, marked with a colored paper to distinguish them from explosives, satisfies most blasting requirements. Tamcot is a fibrous noncombustible material made up into stemming plugs 1½ by 6 in. and used in place of ordinary stemming. Mill tailings and cuttings from drill holes are used as stemming in churn-drill holes.

Blasting Warnings.—Before setting off a blast it is necessary to warn all persons away, to block off all approaches, and to signal by whistle or other means. In open-pit operations, blasting areas are marked off by flags and warning signals are given before blasting.

Bombs.—Bombs consist of a bundle of high-explosive cartridges within which is placed a primer cartridge with electric blasting cap. The bundle is wrapped in burlap and securely tied to lath or sectional ¾-in. pipe, by which it is raised into position. The support is braced and the charge is fired from a safe position. Bombs are used principally for clearing a hung-up chute or rock transfer.

Blockholing and Mudcapping.—Large boulders are drilled and blasted by small charges (blockholing), or they may be smashed by exploding a charge on top of the rock, the explosive being covered by a cap of mud (mudcapping). Fuse and blasting caps or electric detonators are used for the work. Underground, blockholing is required to break down pieces too large to pass grizzlies or chutes. In open-pit operations, pieces too large to pass through the dipper of the shovel are cast aside and reduced by blockholing. Blockholing gives a more efficient use of explosives than does mudcapping.

Improvements in contact blasting designed to avoid both blockholing and mudcapping have been made at the property of the National Tunnel and Mines Company, Tooele, Utah, by W. T. Warren. The idea stems from the "Monroe effect" obtained in demolition operations by concentrated charges. The explosive, a 45 per cent gelatin, is packed into a tin container, resembling a small brimless derby, in such a way as to leave a deep concavity in the area of the charge next to the rock face. An electric blasting cap is placed at the apex of the charge. The container is placed against the face of rock or boulder to be blasted. Medium sized boulders, 45 cu. ft., are said to have been completely broken by a 0.7-lb. charge, and 240 cu. ft. boulders were broken by 4½-lb. charges, the largest charges used to date. Blocked chutes and finger raises have been successfully cleared by similar charges.

Coyote-hole Blasting.—Coyote holes are small tunnels driven into a bank and connected with blasting chambers winged at right angles to the entrance tunnel. Charges of explosive are placed in the chambers along with priming charges. Only high banks should be blasted in this manner. Electric blasting caps are used in the priming charges, and great care is necessary to protect lead wires from damage during the preparation of the blast. Often two independent circuits are used to ensure detonation; sometimes two priming charges are placed in each chamber. Black powder is loaded in unopened cans. Small-diameter cartridges are loaded in unopened cases; large-diameter cartridges are removed from their cases and stacked upon their sides. Primer charges are placed well within the pile of explosives. Stemming material is loaded into the T formed by the junction of the entrance tunnel and the wings, the entrance tunnel being also filled with stemming. Coyote-hole blasting has been superseded to a considerable extent by well-hole blasting, and is now only used where the topography prevents the use of churn drills or where such equipment is not available.

Spitting Fuse.—Many ways are used for igniting fuse. The candle snuff has disappeared, but the acetylene-lamp flame finds limited use. The simplest way to ignite a single fuse is to split the end about ¾ in. down, insert a match head, and strike the match head upon the box; a spurt of flame indicates that the fuse has been ignited. At Butte, 6 to 15 fuses are bunched, tied or taped together, cut evenly on the ends, and ignited by the flame from a short length of fuse notched at 2-in. intervals and previously ignited. After the bunch of fuses has been thoroughly ignited it is held against the face of the second bunch and rotated so as to ignite it thoroughly. In South African gold mines a fuse lighter or Tchisa stick is used for fuse lighting. It consists of a paper shell ¼ in. in diameter by 8 in. in length, containing a mixture of oxidizers and combustibles, which when ignited produces a strong, pointed flame. The stick burns for 4 min. The safety-fuse match lighter consists of a short paper tube that is slipped over the end of a freshly cut fuse. One end of the tube is coated with a composition similar to that used for safety matches and can be ignited by striking upon the side of a safety matchbox. The pull-wire fuse lighter is a paper tube containing an ignition compound and a pull-wire ignition head. After it is slipped over the fuse end, the wire is pulled and the ignition compound lights the fuse. The lead-tube spitter lighter and the hot-wire lighter are other devices. The master fuse lighter, used in sequence or rotation firing, is a waterproofed open-ended paper shell containing an ignition compound in the closed end and a flexible rubber cover on the open end. From two to six freshly cut fuse ends can be slipped through the cover, together with a pilot fuse, until contact is made with the ignition compound. Spitting the pilot fuse ignites all the fuses simultaneously.

Nomenclature of Blastholes.—Position, direction, and function of blastholes are indicated by a not too precise nomenclature that is often local in origin. Uppers and lowers require no definition; direction may be vertically up or down or at an angle. Breast holes are holes in a vertical face and are usually horizontal. Back holes are in a more or less horizontal face above the driller. Cut holes are usually at an angle to the face and are designed to make the initial break in blasting; they are the first to be fired in sequence or rotation blasting. Pyramidal cut refers to three or four cut holes that form the corners of a pyramid and usually meet at the apex. A V or wedge cut is formed by two lines of holes that enclose the wedge. A double V cut consists of an inner wedge cut, as well as an outer. A draw cut is a wedge cut at the side of shaft, winze, or raise section or at the lower half or upper half of a drift section. A diamond cut is similar to a pyramidal cut. A sump cut is sometimes used in shaft sinking, one-half of the shaft section being broken by a wedge cut at the end and the other half-section offset by half this depth of the round similarly broken. A slabbing cut consists of from two to four steeply angled holes not symmetrically placed. A burn cut (shatter cut, woodchuck cut) refers to a group of horizontal holes in the central part of a face that break out a cylindrical core, some holes being charged and some being left empty.

"Relievers" is a term applied to the second and often to a third group of holes surrounding the cut holes and breaking in sequence to them. "Trimmers" applies to the holes close to the periphery of the face. They have the function of breaking to the limits of the cross section, and are the last of the series to be blasted. "Lifters" refers to the bottom row of holes on a section. They may be the last holes to be blasted. They are often more heavily charged so as to throw back the muck.

"Burden" refers to the normal distance from the charge in a hole to a free face or to a break established by a previously blasted group of holes (sequence blasting). "Overbreak" (backbreak) refers to the blasting of a face beyond the lines of the intended section. Underbreak is a face that in one or more portions falls short of completing the section. Pop shots are charges placed in short drill holes to complete breaking to the section lines. "Toe holes" refers to a line of holes at the base or toe of a bench in open-pit mining.

Concentrated charges are charges in a chamber formed in the lower portion of the drill hole. Column charges are so placed in a churndrill hole as to form an unbroken column. "Deck loads" refers to separate charges placed at intervals in the drill hole above the principal charge and separated by lengths of stemming. Spaced charges are two or more portions of explosive separated by spacers in a drill hole, the purpose being to distribute the blast along the length of the drill hole. Spacers of wood (objectionable) or pottery (preferable) are 8 to 10 in. in length and about one-half the diameter of the borehole. By the use of high-count or low-density cartridges a charge may be distributed over a greater length of drill hole and the use of spacers avoided. In cushioned blasting, an air space is left by introducing a copper-wire spacer before the stemming or by using cartridges of smaller diameter than the blasthole.

Misfires are charges that have failed to explode. They are the source of many accidents. Cutoff holes are holes that have their upper portions blown off by neighboring blasts and often contain unexploded charges. After a face has been blasted and a safe time has elapsed, it is examined for sockets and misfires. The muck pile should also be examined for unexploded cartridges derived from cutoff holes. Only experienced blasters should be permitted to handle misfires. Where electric blasting caps have been used, the blasting circuit is disconnected before approaching the face and, if leg wires are accessible, the cap is tested with a circuit tester. If it shows normal condition, the leg wires are connected up and an attempt made to explode the charge. Misfires are sometimes handled by removal of the stemming by means of compressed air or water discharged from a hose; a fresh primer is inserted and the charge is fired. Unexploded charges in sockets are handled in the same way. Attention to all details of blasting reduces the liability of misfires. In open-pit operations it is especially necessary to maintain records of charges in each drill hole, position of deck charges, and stemming heights, as this information may be helpful in handling a misfire.

Throw, Concussion, and Seismic Effect.—In hard-rock blasting, pyramidal and wedge cuts throw broken rock a considerable distance, sometimes 50 to 100 ft. in horizontal development, and in shaft sinking timber sets may be greatly damaged. In soft and medium-hard rock the throw is much less. The burn cut has the advantage of producing little or no throw. Slabbing cuts are also used to reduce throw. Excessive charges may produce throw. Lifters, the last holes in a round to explode, are often the most heavily charged, so as to throw broken rock back from the face. Safe distances when firing charges are 1,000 ft. or more from a face, or around a turn or two in connected workings.

In bench blasting there is a moderate throw, even with well-proportioned charges. Widespread or scattering throw is usually due to excessive charges. In many open-pit mines the first shovel cut gathers the throw of the blast and loads it out, using the tracks in place; on completion of the scatter cut, the tracks are shifted and a second cut completes the cleanup of the bench.

Concussion or air blast is greatest near a blast. Damage to structures, unless they are abnormally close, is nil. Every blast sets up vibrations, local in character, in the rock mass. Large charges may set up vibrations that may be detected at a considerable distance by seismographs. J. R. Thorn and S. L. Windes (*Bulletin* 442, U.S. Bureau of Mines) state that customary vibrational disturbances from quarry blasting result in displacements and frequencies that represent accelerations of about 0.01 of gravity. Ordinary quarry blasting does not produce damage to buildings or other structures.

CHAPTER VI

BLASTING ROCK

Blasting.—The explosion of a relatively small volume of explosive within a rock mass releases gaseous products at high temperatures and high pressures. The shock or blow thus places the rock mass under great stress, and displacement and fracture may take place in the direction of least resistance if there is no opportunity for escape. To displace and fracture a given rock mass, the weight of explosive must be in proportion to the volume of the mass. Experience as well as experiment has determined the weight-to-volume ratios for different explosives and for varying conditions. This weight of explosive may be concentrated in a single charge, as in coyote-hole blasting, or it may be distributed in a number of small charges in as many blastholes, so placed as to equalize the proportion of burden on each charge.

Lines of least resistance, the number of free faces, whether the exposed rock face is normal to the line of least resistance, the depth of the blastholes, the strength of the rock mass, and the kind of explosive are all considered by the blaster. The least favorable condition is a single free face, such as the bottom of a shaft or the face of a heading. The most favorable is a detached rock mass. In the former more explosive is required to break a given volume of rock than in the latter, where a relatively small weight will shatter the mass.

Under some conditions displacement and fracture may result in a high proportion of large rock masses; with a greater weight of explosive distributed in a greater number of charges and by the use of a high explosive the proportion of large masses may be reduced. That is, better fragmentation is secured. Large masses must be reduced by further blasting, which is called secondary blasting. Good fragmentation in the first blasting reduces the amount of secondary blasting.

Bench Blasting.—Bench blasting as here considered deals with blasting in open-pit mines and in rock quarries other than dimension- or building-stone quarries. The requirement is to break the rock small enough to be conveniently handled by the steam shovel or by hand. The subsequent treatment of the rock must be considered. If it is to be crushed in rock breakers, the maximum-sized piece that can be handled by the rock breaker in question determines the size limit of the product. A blast will result in a greater or less amount of oversize material, and this must be reduced by blockholing and blasting or by sledging. The blaster endeavors to produce the minimum amount of oversize without at the same time producing an excessive amount of fine material. Good judgment in spacing holes and in the selection of the kind and amount of explosive is necessary to accomplish this end.

In Figs. 58*a*, *b*, *c*, *d*, and *e* the methods established by mining practice are shown. In *a* a row of drill holes is placed parallel with the crest of the bench and spaced a distance apart equal to the line of least resistance. The depth of the hole is made somewhat greater than the height of the bench. The line of least resistance *W* may equal the depth of the drill hole *D* or may be some fractional part of *D* like 0.75*D*, 0.5*D*, 0.33*D*, or 0.25*D*. If chamber blasting is employed the first and second values are suitable. In chamber blasting the bottom of the drill hole is enlarged by repeated blasts of a high explosive. First several sticks of powder are blasted, using little or no tamping. Then, after an interval for the drill hole to cool off, more powder is crowded into the bottom and the chamber is further enlarged. Two or three charges may be necessary before a chamber of sufficient size is obtained. Where it is desired to break the rock to small size, the third and fourth values are used. The weight of the powder charge and the size of the drill hole also have their

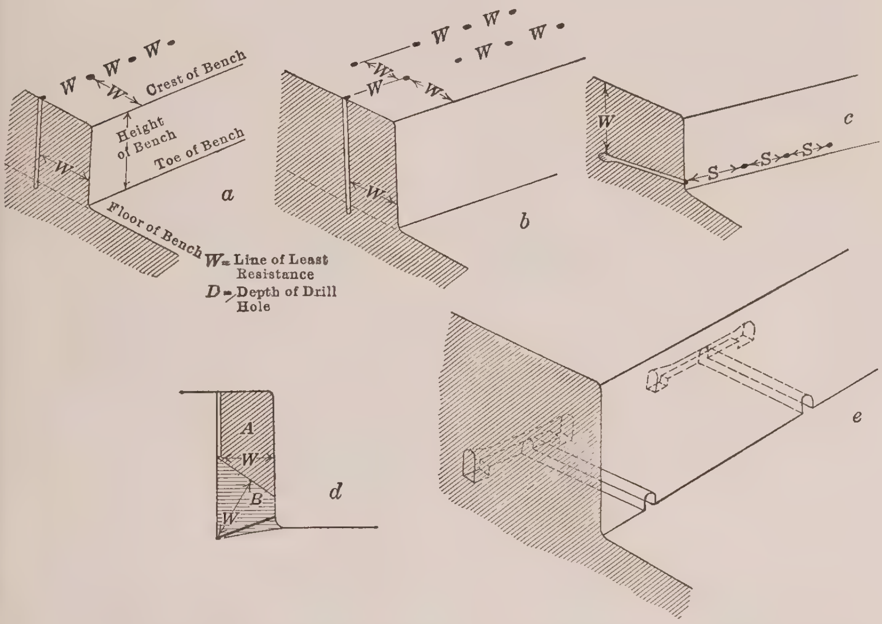


FIG. 58.—Distribution of blasting charges used in bench blasting. In *c*, *S* is the distance between drill holes; in *d*, *A* is broken by one hole and *B* by the toe hole.

influence. The depth of the charge in a drill hole where chambering is not made use of is given by the equation

$$\text{Charge length} = D - 1.2W$$

This allows a length of stemming somewhat greater than the length of the line of least resistance. The resistance both at right angles to the drill hole and in the direction of its length is thus equalized. By making W smaller than D a sufficient charge of powder can be placed in a drill hole of small diameter. For example, assume that $W = \frac{D}{2}$, and take a drill hole 30 ft. in depth. The charge length would equal 12 ft. Assume that 135 lb. of charge would be necessary. A drill hole 5 in. in diameter would permit this charge (assuming the specific gravity of explosive to be 1.2). If W were greater than 15 ft., more explosive would be required and would have to be placed in a shorter charging chamber, thus necessitating a drill hole of larger diameter.

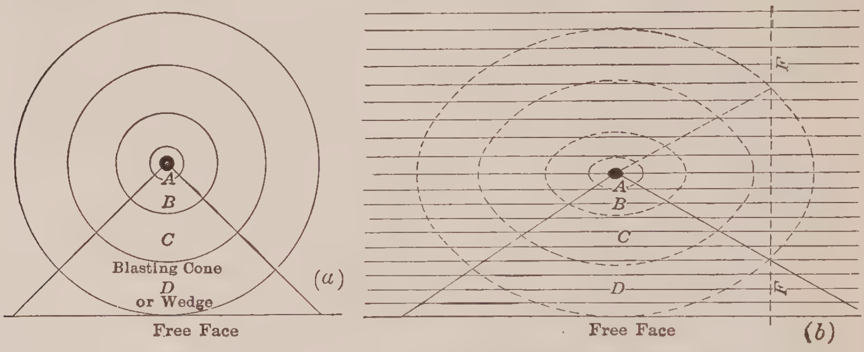


FIG. 59.—Blasting cone; (a) homogeneous rock, (b) stratified rock.

The interval between the drill holes along the length of the benches ranges from W to $2W$. The degree to which the rock is to be broken and the cohesiveness of the rock determine the interval. Where tough, solid rock is to be broken small, the first value is suitable, whereas if large masses are required the second value would be used. In order to understand the effect of simultaneously blasting a number of drill holes it is necessary to study the blasting cone in detail. In Fig. 59*a* is shown a section of a blasting cone in homogeneous rock. The concentric rings represent imaginary shells in a sphere of rock that represents the mass of rock affected by the blast. In the innermost shell the rock is broken to a fine powder; in the second, *B*, it is broken into small fragments; in the third, *C*, it is broken into coarser fragments, while in the fourth, *D*, the fragments are considerably larger. There is no sharp division surface between the shells. Displacement is possible only in the direction of the free face, and in this direction a cone-shaped mass is displaced and the fragments are free to fall apart. The remainder of the volume of the sphere consists of a compact mosaic. No movement being possible in this portion, it is reasonable to expect that the degree of fracturing is somewhat less than in the displaced cone. The diameter of the sphere depends upon the strength of the rock and the weight of explosive used.

In Fig. 59*b* the conditions in a nonhomogeneous rock mass are shown. In this case the rock volume affected is a flattened sphere. The flattening is transverse to the lines of stratification. As in the former case, concentric spheroidal shells can be imagined. The displaced cones are either less or greater than a right-angle cone, depending on whether the free face is parallel with or at right angles to the stratification planes. Both figures also illustrate the conditions where the charge of explosive is distributed in the form of a long cylinder. The cones become wedges in these cases.

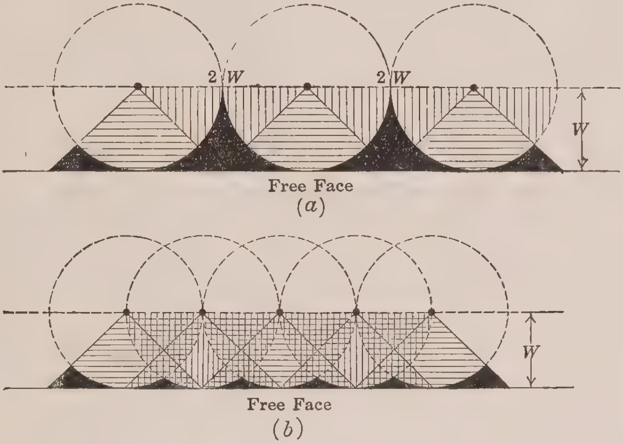


FIG. 60.—Simultaneous blasting; (*a*) holes $2W$ apart, (*b*) holes W apart.

Figures 60*a* and *b* illustrate the conditions resulting from simultaneous blasts. In *a* the boreholes are spaced twice the length of the line of least resistance, while in *b* the spacing is the length of the line of least resistance. The blasting spheres are tangent in *a*; in *b* they overlap. In both cases the hatched and shaded portions indicate the section of the mass of rock affected. The heavily shaded areas indicate the rock masses which are displaced, but which are without the sphere or cylinder of the greatest fracturing and hence are displaced in relatively large masses. The overlapping of the spheres of greatest fracturing in Fig. 60*b* indicates how these rock masses are reduced in size and how certain portions of the rock mass between the blasts are subjected to twice the amount of fracturing as compared with case *a*.

The approximate volumes of rock broken by a single blast under varying conditions and also by a number of simultaneous blasts are given in Table 21. Two free faces are assumed.

TABLE 21

Value of W	Volume broken by single blast	Volume broken by (n) blasts	
		Spacing W	Spacing $2W$
$0.75D$	$0.56\ D^3$	$0.56\ nD^3$	$0.56\ (n + (n - 1))D^3$
$0.50D$	$0.25\ D^3$	$0.25\ nD^3$	$0.25\ (n + (n - 1))D^3$
$0.33D$	$0.11\ D^3$	$0.11\ nD^3$	$0.11\ (n + (n - 1))D^3$
$0.25D$	$0.062D^3$	$0.062nD^3$	$0.062(n + (n - 1))D^3$

D, depth of drill hole; W, line of least resistance; n, number of holes.

The calculation of the approximate weight of the explosive required can be obtained by multiplying the volume to be broken by the powder ratio, which is the weight of the explosive in pounds required to break 1 cu. ft. of rock. It is determined by a number of trial shots carried out under as nearly as possible the same conditions as the blasting to be done. Sometimes "standard rounds" are determined by trial for different kinds of mining work. In the experimental work the depth of holes, speed of drilling, and the quantity and kind of explosive are determined for the several conditions. In Table 22 different powder ratios have been assumed and the powder charges calculated for different drill holes.

The first method (Fig. 58*a*) has been considered in detail for the reason that it is the most frequently occurring case and the other methods are close parallels. In method Fig. 58*b* two lines of drill holes are placed, the spacing in both directions being shown as equal to the line of least resistance. The holes in the back row are placed between the holes in the front row. The result of spacing in this way is to obtain a smaller amount of oversize product. A somewhat greater efficiency of the explosive favors this arrangement. Method Fig. 58*c* is used in soft rock or ore where the holes can be readily dug with a bar. It is frequently termed gopher-hole blasting. Method Fig. 58*d* is used where a high bench is worked and the economical depth of the drill hole is limited. A 50-ft. bench could be broken by two 25-ft. holes. The rock broken by each hole is shown by the shaded portions *A* and *B*. Method Fig. 58*e* is used for blasting benches of considerably greater height than can be economically drilled. Benches 100 to 200 ft. in height can be broken. The procedure is to run a crosscut into the toe of the bank and from the end of the crosscut to extend two short drifts terminating in powder chambers.

TABLE 22

D, feet	V in cubic feet	Weight of powder in pounds per charge				
		Powder ratios, pounds per cubic foot				
		0.05	0.04	0.03	0.02	0.01
50	A 31,250	1,562.0	1,250.0	938.0	625.0	312.5
	B 13,888	694.0	556.0	417.0	278.0	139.0
	C 7,812	391.0	312.0	234.0	156.0	78.0
40	A 16,000	800.0	640.0	480.0	320.0	160.0
	B 7,111	355.0	284.0	213.0	142.0	71.0
	C 4,000	200.0	160.0	120.0	80.0	40.0
30	A 6,750	338.0	272.0	203.0	135.0	68.0
	B 3,000	150.0	120.0	90.0	60.0	30.0
	C 1,687	84.0	68.0	50.0	34.0	17.0
20	A 2,000	100.0	80.0	60.0	40.0	20.0
	B 888	44.0	36.0	27.0	18.0	9.0
	C 500	40.0	20.0	15.0	10.0	5.0
10	A 250	13.0	10.0	8.0	5.0	2.5
	B 111	6.0	4.4	3.0	2.0	1.1
	C 62	3.0	2.48	2.0	1.2	0.62
5	A 31	1.65	1.24	0.93	0.64	0.31
	B 14	0.70	0.56	0.42	0.28	0.14
	C 8	0.40	0.32	0.24	0.15	0.08

D, depth of borehole; W, line of least resistance; V, volume broken by single shot in cubic feet. For A, $W = \frac{D}{2}$; for B, $W = \frac{D}{3}$; for C, $W = \frac{D}{4}$.

Black powder has generally been displaced by low-velocity explosives and bulk or bag powders. In most open-pit mines churn-drill holes are used for bench blasting; coyote-hole blasting is disappearing or is used where conditions for churn drilling are unfavorable. The 9- and 12-in. churn-drill holes enable sufficiently large charges to be placed without chambering. Column charging or column plus one or more deck loads (Fig. 61) is customary practice, the explosive being distributed in proportion to the burden along the depth of the hole. Where bench slopes are such as to place too great a burden upon the toe, supplementary toe holes are necessary. Banks are usually 35 to 40 ft. in height. Two or more parallel lines of blastholes may be necessary to break the width of the cut, which may range from 35 to 65 ft. Groups of holes are simultaneously exploded, and banks are blasted well in advance of shovel operations as little as a month and often a season in advance. Table 23 gives weights of explosive per foot of drill hole.

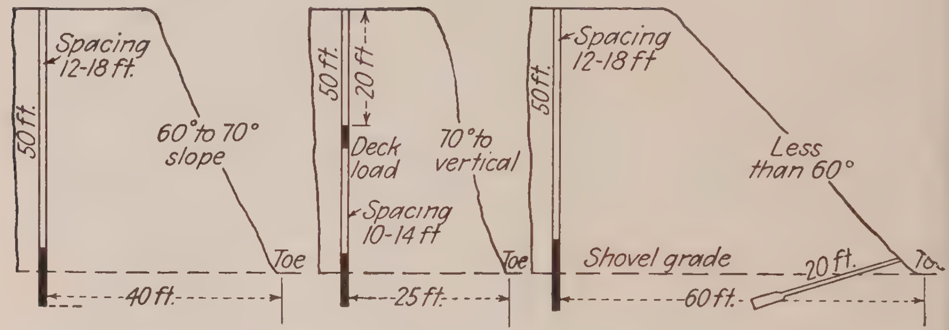


FIG. 61.—Bench heights and hole spacing at New Cornelia, Ariz. (*A.I.M.E.*)

At New Cornelia (Arizona) machine-drill holes are used upon 20-ft. benches; holes are 24 ft. in depth and 12 to 16 ft. back from the toe of the bench, spaced 12 ft. apart in soft ground, 11 ft. in medium, 9 ft. in hard ground, and 8 ft. in very hard. They are sprung once in soft and medium, twice in hard ground, and three times in very hard rock. Mean area per drill hole, assuming an 80-deg. slope of bank, is 120 to 168 sq. ft. in soft rock, 110 to 154 in medium, 90 to 126 in hard, and 80 to 112 sq. ft.

TABLE 23.—WEIGHT OF EXPLOSIVE PER FOOT OF DRILL HOLE

Diameter drill hole, in.	Vol of 1 ft., cu. in.	Weight of explosive in 1 ft. drill hole				
		Black powder	Permissible	Ammonia dynamite	Ammonia gelatin	Gelatin
1.25	14.7	0.5	0.56	0.68	0.87	0.71
1.50	21.2	0.8	0.81	0.98	1.25	1.02
2.00	37.7	1.4	1.43	1.73	2.22	1.82
3.00	84.8	3.2	.....	3.90	4.99	4.09
5.00	235.6	8.7	.....	10.83	13.90	11.34
6.00	339.6	12.6	.....	15.62	20.03	16.40
8.00	603.2	22.3	.....	27.74	35.59	29.13
9.00	763.4	28.2	.....	35.19	45.04	36.87
12.00	1357.2	50.2	.....	62.43	80.07	65.55
Weight, lb. per cu. in.	......	0.037	0.038[a]	0.046	0.059[b]	0.048

[a] 133 cartridges per 50-lb. case; a 256 count gives 0.02 lb. per cu. in.

[b] 90 per cent explosive; for 30 per cent the weight is 0.046. Large cartridges of 40 and 60 per cent ammonia gelatin weigh 0.0552 per cu. in.; other ammonia semigelatins, 30 and 40 per cent, weigh 0.036 lb per cu. in. Nitromon in cans, grade A (75 per cent), weighs 0.049 lb. per cu. in.; grade D (40 per cent) weighs 0.034 lb. per cu. in.

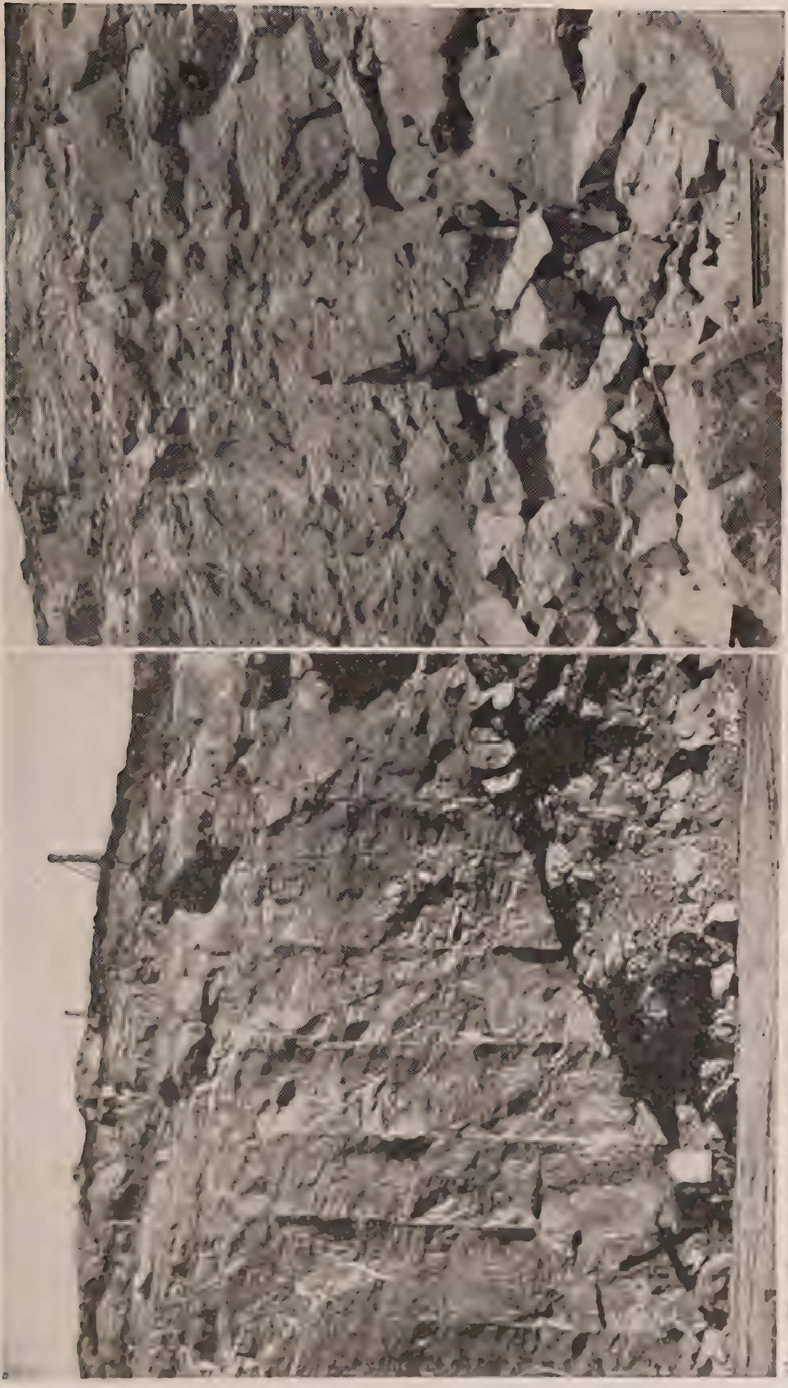


FIG. 62.—Bench blasting by churn-drill holes in granite quarry; note size of rock fragments on right comparing with figures of men. (*Bucyrus-Erie Co.*)

in extra hard, the foregoing quantities also indicating cubic feet per foot of drill hole. Nine-inch churn-drill holes for 40-ft. benches are 46 ft. in depth and 23 to 35 ft. from toe of bench. They are spaced 14 ft. apart for soft and medium rocks, 12 ft. for hard, and 10 ft. for extra hard. The computed mean area per drill hole, assuming an 80-deg. bank, is 231 to 329 sq. ft. for soft and medium rock, 198 to 282 for hard, and 165 to 235 for extra-hard rock. For 60-ft. benches, 68-ft. drill holes 30 to 50 ft. from the toe are spaced 18 ft. apart in soft ground, 16 in medium, 14 in hard, and 12 in extra-hard rock. Computed mean areas per drill hole, assuming an 80-deg. bank slope, are 441 to 711 sq. ft. for soft ground, 392 to 632 for medium, 343 to 553 for hard, and 294 to 474 for extra-hard rock. The softest rock encountered is described by H. H. Angst and R. A. Cochrane, from whose paper the foregoing was derived, as leached monzonite capping; diorite and rhyolite formations vary from extremely hard to very soft, and quartz monzonite is extremely hard and siliceous in character.

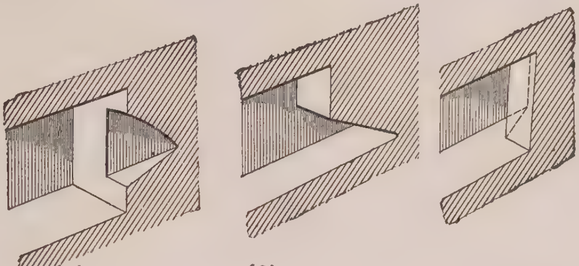
At the Utah Copper open pit, 70-ft. benches are broken by toe holes 23 ft. in depth and spaced 15 ft. apart; 17 holes, carrying a burden of some 25,000 tons, are blasted simultaneously. At Chino a 50-ft. bank is broken by churn-drill holes 10 ft. back from bank edge, spaced 20 to 30 ft. and supplemented by 20- to 22-ft. toe holes 15 to 20 ft. apart. In coal strip pits the burden placed upon drill holes ranges from 10 to 40 cu. yd. per foot of drill hole; in limestone quarries the range is from 6 to 20 cu. yd. per foot of churn-drill hole. On the Mesabi Range churn-drill holes are drilled 5 ft. below the bench grade; benches are 25 to 35 ft. in height; holes are spaced 16 to 35 ft. apart, and rows of holes are 15 to 20 ft. apart.

Figure 62 is from photographs of a large granite quarry in Southern California in which a high bench was carried, the breaking being by churn-drill holes and blasting. The left picture shows the completing of the cleanup of a blast and the right-hand picture the coarse broken rock resulting from the blasting of a row of churn-drill holes.

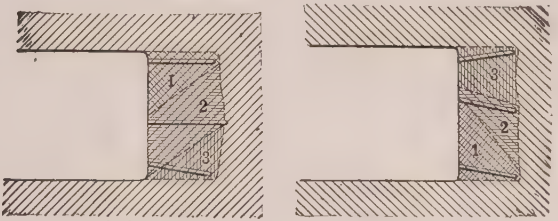
Mine Workings.—Drift, crosscut, tunnel, adit, raise, and shaft present a similar problem in breaking in that one free face is attacked. Many patterns of rounds are found in practice. The number of holes per round and their depth, angle, and position vary with the type of rock broken and the dimensions of the face, but in most mining operations standardization in repetitive work results in greater speed and lower costs. Fragmentation suitable for mechanical loading appliances is furthered by a large number of drill holes and the selection of a suitable explosive.

The greater the area of the section, the fewer drill holes; the greater the depth of the round, the deeper the drill holes and the greater the explosive charges necessary; the smaller the section, other factors being equal, the greater the amount of explosive per cubic foot broken.

Most rounds initiate the break by a pyramid, diamond, V, or burn cut; other holes break to this in succession, the trimmer and lifter holes completing the section. The V cut may be at the center, on the side, bottom, or top of the face. Figure 63 indicates the pyramid cut at *a* and the V or wedge cut at *b* and *c*. The angle of the pyramid or wedge should be such as to make the bottom of the holes forming it deeper than the breaking depth of the round. The double V and double pyramid cuts may be advantageous for a very deep round in very hard rock; they involve a shallower wedge or pyramid within an outer wedge or pyramid. The simplest V cut consists of two angle holes meeting at a point just beyond the depth of the round; other holes are so placed as to be readily drilled and also to break to the cut holes. Patterns of rounds are tried out in a given working until one has been found that best meets the needs of the situation. The depth of the round is usually taken as three-quarters of the maximum dimension of the face; thus the 5- by 7-ft. drift is given a 5-ft. depth of round. A burn cut, on the other hand, is planned to give a greater depth, from 1 to 1½ times the greatest section dimension, but about 12 ft. seems to be the limit. Figure 64 *a* illustrates a pyramid cut for hard rock; *b* a V or wedge cut for medium rock in a small drift; *c* the heading method of driving a large tunnel in medium rock, the heading being broken by a V cut and the bench by vertical holes; *d* and *e*



(*a*) Center Cut (*b*) Bottom Cut (*c*) Side Cut



(*d*) (*e*)

FIG. 63.—Position of initial break in tunnel blasting.

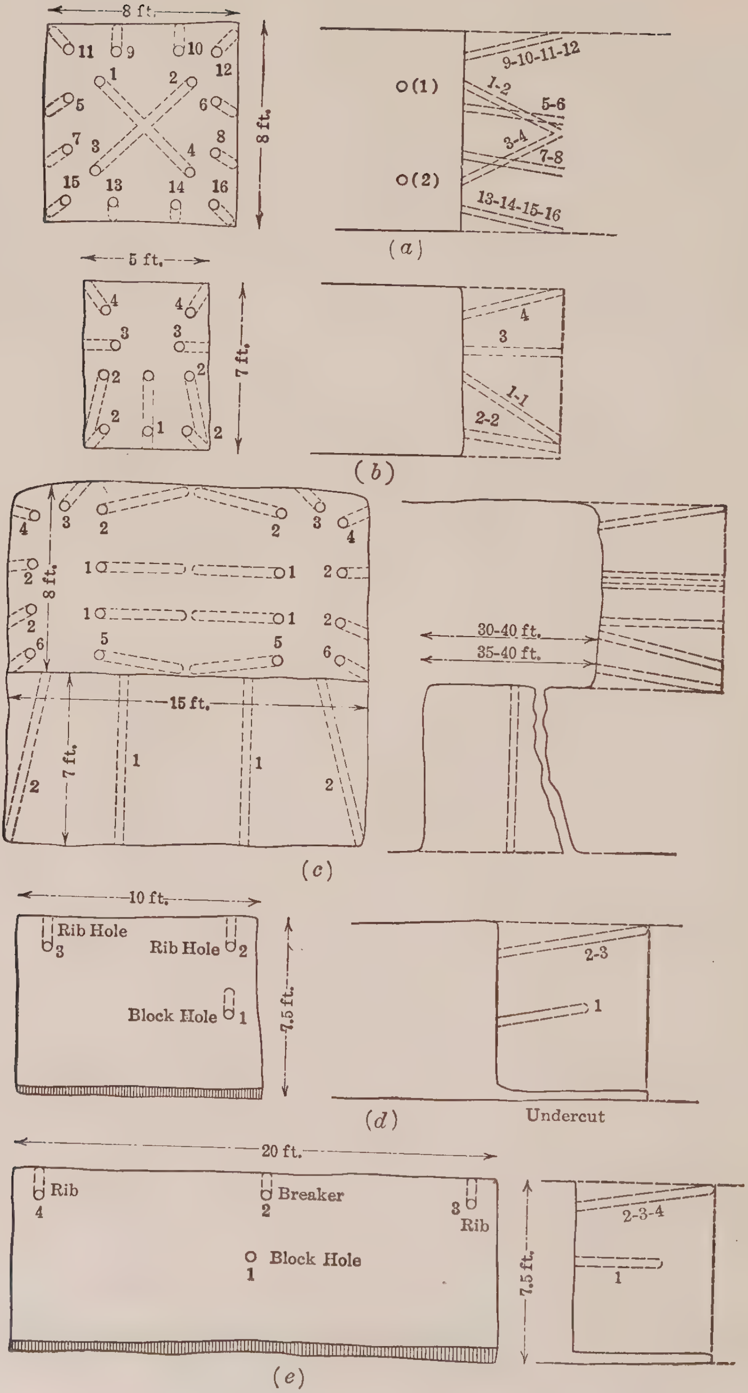
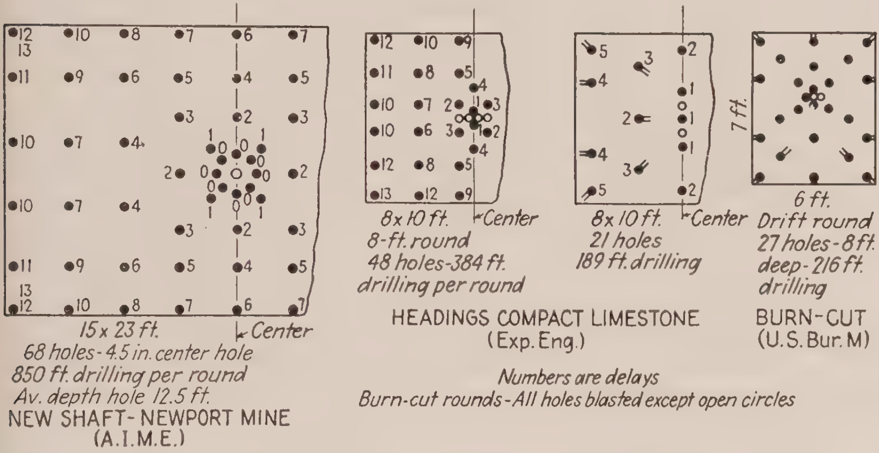


FIG. 64.—Examples of drill rounds.

are examples from coal mining where a deep undercut establishes two free faces. Examples of burn or shatter cuts are given in Fig. 65.

In tunnel and adit work in hard rock, sections range from 100 to 200 sq. ft. in area, with an area per drill hole of 3.5 to 5 sq. ft.; sections ranging from 50 to 100 sq. ft. have an area per drill hole of 2.5 to 3.5 sq. ft. At the Stanislaus tunnel (California) (Fig. 66), with a section of 116 sq. ft. in granite, the ratio ranged from 2.8 to 3.2 sq. ft. area per drill hole; the Carlton tunnel, with an area of 105 sq. ft., gave a ratio of 3 sq. ft.; the Elton, 4.5 sq. ft. Small drifts of 30 to 50 sq. ft. require ratios of from 2.5 to 3 sq. ft. per drill hole in hard rock, 4 to 5 sq. ft. in medium, and up to 7 to 8 sq. ft. in easily broken rock.



15x23 ft.
68 holes-4.5 in. center hole
850 ft. drilling per round
Av. depth hole 12.5 ft.
NEW SHAFT-NEWPORT MINE
(A.I.M.E.)

8x10 ft.
8-ft. round
48 holes-384 ft.
drilling per round

8x10 ft.
21 holes
189 ft. drilling

HEADINGS COMPACT LIMESTONE
(Exp. Eng.)

Drift round
27 holes-8 ft.
deep-216 ft.
drilling
BURN-CUT
(U.S. Bur. M)

Numbers are delays
Burn-cut rounds-All holes blasted except open circles

Fig. 65.—Examples of burn-cut rounds.

Cubic feet of rock broken per foot of drill hole is another interesting ratio. In hard rocks the ratio is from 2.1 to 2.5 for small drifts of 30 to 60 sq. ft.; larger sections, 80 to 100 sq. ft., give a ratio of 5 to 8 in medium rocks per foot of drill hole. The Stanislaus gave a ratio of 2.16 to 2.43 cu. ft. per foot of drill hole in hard granite.

In vertical work the round pattern is much the same as in horizontal. The V cut and double V cuts are principally used and the burn cut (Fig. 65) is now more or less common. Figure 67 shows a double V-cut round, a sumping round, and a round for a shaft of circular section. Figure 68 shows two round patterns, one taking advantage of a slip plane and the other a double V cut, both rounds being drilled by hand-held sinkers. The numbers used in the figure indicate the sequence of firing. Many shaft rounds are designed with a V cut at one end; a sump is maintained by alternately taking out one-half of the shaft section. In the burn cut all holes are vertical. An area in the center of the shaft is drilled with two rings of eight holes each, the inner ring being a foot in diameter and the

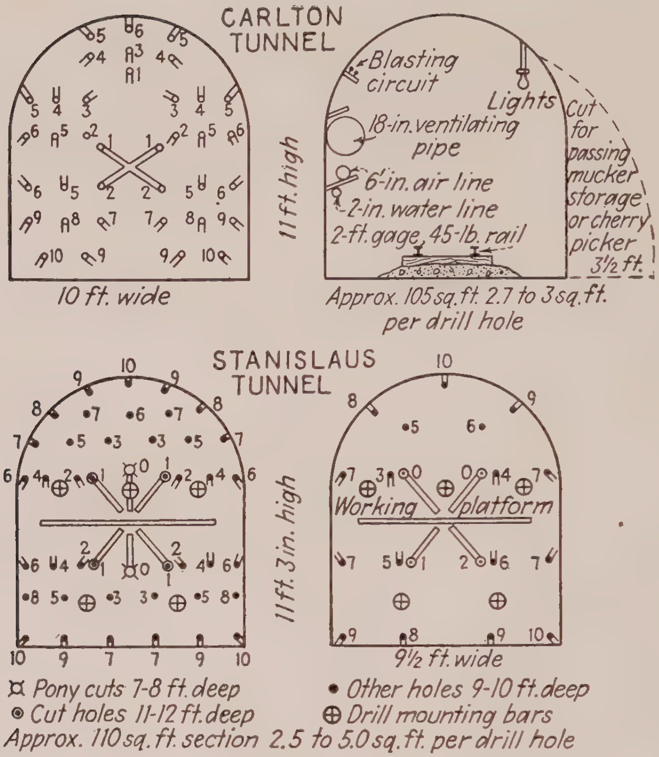


FIG. 66.—Carlton and Stanislaus tunnel rounds. (*Eng. News-Record and Construction Methods.*)

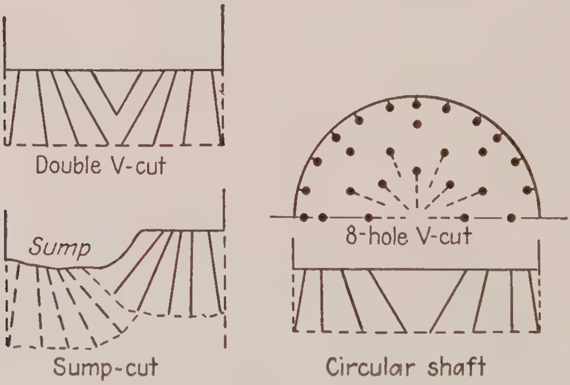


FIG. 67.—Examples of shaft rounds.

outer 18 to 24 in., a large-diameter hole (3 to 4 in.) being also drilled in the center of the ring. All other holes are in parallel rows. The burn cut is favored for shafts of large section. Holes for a circular shaft section are drilled in rings about an eight-hole pyramidal-cut group in the center area, relievers being staggered with respect to the cut holes and an outer row of trimmers being drilled about 2.5 ft. apart and close to the periphery of the section. In raising, the V cut is placed at one end of the section above the waste-rock chute, the remaining holes in rows breaking successively to the cut.

Drill-hole ratios in shafts of large section range from 3.8 sq. ft. in hard rocks to 5.5 to 9.6 sq. ft. in medium rock; the ratio for smaller sections in hard rock is 3.5 sq. ft. The cubic-foot ratio ranges from 2 to 4 in hard rock up to 5 to 9 cu. ft. per foot of drill hole in medium rock. The ratio in raising is 2 to 4 sq. ft. in hard rock and 3 to 5 sq. ft. in medium rock; in small sections 16 to 40 sq. ft. in area the range is 2 to 3 sq. ft. and in large cross sections 5 to 6 sq. ft. per foot of drill hole. The cubic-foot ratio is 1.5 to 3 cu. ft. per foot of drill hole for small sections (16 to 54 sq. ft.), and 4 to 5.5 cu. ft. for sections 60 to 100 sq. ft. in area.

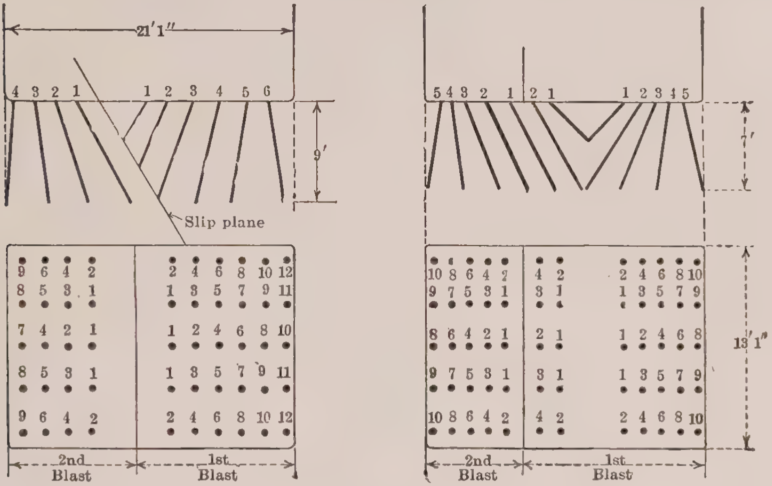
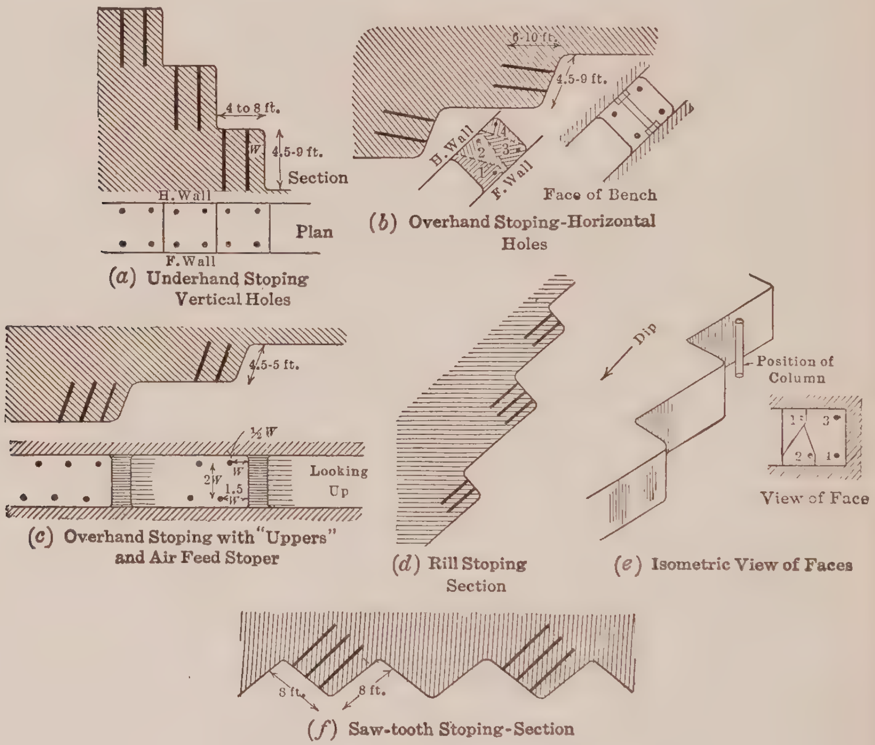


FIG. 68.—Distribution of drill holes in shaft sinking. (*Trans. L.S.M.I.*)

In stoping (Fig. 69), the initial break is sometimes by V cut; the other holes are normal or slightly inclined to the breast or back and break to the cut. The area of stope face per drill hole is from 8 to 12 sq. ft. and the break 8 to 12 cu. ft. per foot of drill hole. Drill holes are from 6 to 20 ft. deep; a common depth is 10 ft. Where long-hole drilling is practiced or where the diamond drill is used, stoping-hole depths may go up to 20 to 70 ft. or more. Most stoping is geometrically similiar to bench blasting but on a smaller scale. Sublevel stoping at Noranda (Canada) required holes 11 to 20 ft. in depth in sulphide ores, the burden 33.3 cu. ft. per foot of drill hole; diamond drilling in rhyolite stopes required holes 16 to 65 ft. in depth with a burden of 52 cu. ft. per foot of drill hole.

The Burn Cut in Drifts and Crosscuts.—For some 10 years the burn cut or shatter cut has attracted attention, but no extended studies of the principles involved appear to have been made. In some mines experimental rounds have been tried out and the most successful adopted.



(*a*) Underhand Stoping Vertical Holes

(*b*) Overhand Stoping-Horizontal Holes

(*c*) Overhand Stoping with "Uppers" and Air Feed Stoper

(*d*) Rill Stoping Section

(*e*) Isometric View of Faces

(*f*) Saw-tooth Stoping-Section

FIG. 69.—Distribution of drill holes in stoping.

The principal objective is an increase in the depth of round, as this means fewer setups, more efficient drilling, and usually faster progress. Another advantage is less throw of the broken rock and less dust. Some rounds are characterized by having all drill holes normal to the face, others by the combination of angled holes and normal holes. Usually the so-called "burn cut" consists of from five to eight holes, of which one or more are left open or not charged. At one time the cut was blasted first and then the remainder of the round was loaded and fired, but present practice is to use delayed exploders and to fire all at one time.

The closely spaced cut holes break together and thus establish a substantial line of weakness to which the relievers may break. In many cases about half of the cut holes are drilled larger in diameter than the remainder. More holes and therefore a greater drill footage is required per round than in the ordinary drift round. The square footage of face per drill hole ranges from 1.66 to 2, and about 1.5 to 2 cu. ft. are broken per foot of drill hole. More explosive relative to volume is usually required.

A. W. Jacobs and T. E. Newkirk, describing the use of the shatter cut at the McIntyre-Porcupine mine for 8- by 8-ft. headings, show in their charts that depth of round slightly exceeded 7 ft.; explosives (40 and 55 per cent) approximated 12 lb. per ft. or 0.2 lb. per cu. ft. The number of holes per round varied from 28 to 31, the drill footage 215 to 238, the square feet of face per drill hole 2 to 2.3, and the cubic feet per drill hole 1.9 to 2.1. The cut, a D round, consisted of six holes in a circle about a center hole, only two being charged and these with 55 per cent explosive; two 6-ft. relievers angled toward the cut holes were also charged with 55 per cent; a pyramid of four holes charged with 55 per cent completed the central zone of the face; seven holes normal to the face occupied an intermediate position, and eight outer holes at corners and mid-points of the section completed the round. The combined four-hole diamond and short two-hole V round is called a combination burn. The average depth of all holes except V holes was 8 ft.

The future development of the burn-cut idea might well be in the use of the broaching bit to break out a line of holes, either in a vertical or horizontal plane, to form a slot to which the outer holes in the round would break. Some of the intermediate holes could advantageously be angled toward the slot.

Explosives Ratios.—The weight of explosive per cubic foot of rock broken is useful in computing blasting charges. Important considerations are the type of explosive and the degree of fragmentation desired. In open-pit and quarry operations, as well as in stoping, smaller explosive ratios are obtained as compared to those required in development workings and tunnel work. The greater the fragmentation, usually the greater becomes the explosive ratio. In addition, the spacing of the holes and the kind of explosive play a part. The energy of the explosive, its position, and the strength of the rock determine how thoroughly it will be broken up by a blast.

Bowie gives an average of 0.0141 lb. per cu. ft., a maximum of 0.023, and a minimum of 0.0095 lb. required in gravel-bank breaking by coyote-hole blasting in hydraulic mining; banks were 50 to 150 ft. in height and Judson (5 per cent) granulated powder was used. Table 24 (after H. H. Angst and R. A. Cochrane) gives the explosive ratios for machine- and

TABLE 24.—EXPLOSIVE RATIOS AT THE NEW CORNELIA OPEN PIT

Formation hardness	Weight of explosive per cu. yd., lb.	Explosive used	Lb. per cu. ft.	Lb. equivalent nitroglycerin	Cu. ft. per lb. nitroglycerin
Very soft	0.50	No. 4 Quarry	0.01850	0.00555	180
Soft	0.625	No. 1 Quarry	0.0231	0.01386	72
Medium	0.75	No. 1 Quarry	0.0280	0.0196	51
Medium-hard	0.875	Ajo special grade	0.0320	0.0128	77.2
Hard	1.00	40% special grade	0.0370	0.0148	68
Extra-hard	1.00	60% special grade	0.0370	0.0222	45
	1.125	60% special grade	0.0463	0.0278	36
Stanislaus tunnel	7.56	40 and 60%	0.2800	0.1377	7.2
	9.64	40 and 60%	0.3570	0.1800	5.5

NOTE.—Velocity ratings of the first six explosives are, in the order listed, 5085, 6234, 6234, 9500, 12,467, and 15,420 ft. per sec.

churn-drill hole blasting at the New Cornelia open-pit copper mine. The explosives given in the table are for dry conditions, but the same ratios were also used with explosives suitable for wet conditions. The ratios required in the Stanislaus tunnel are included for contrast. At other open-pit copper mines ratios range from 0.01 to 0.017 lb. per cu. ft. with with 60 to 65 per cent ammonia dynamite. At iron mines on the Mesabi the ratios are from 0.004 to 0.016 lb. per cu. ft. for ore when 25 to 60 per cent dynamite is used. In coal strip pits ratios are from 0.001 to 0.01 lb. per cu. ft. in shales and limestones. Limestone quarries give 0.013 to 0.022 lb. per cu. ft. with 40 to 50 per cent dynamite.

Underground Explosives Ratios.—In Table 25 explosive ratios for development workings and tunnels are given. The foregoing examples give ratios toward the hard rocks. In shales and slates, explosive ratios are about one-half those suitable for the harder rocks. In contrast to

TABLE 25.—EXPLOSIVES REQUIRED IN ADIT, DRIFT, AND TUNNEL

Example	Area of working, sq. ft.	Lb. explosive per cu. ft.	Explosive, per cent
Continental tunnel	148	0.318	
Carlton tunnel	156	0.266	
Stanislaus tunnel	116	0.28–0.33	40 and 60
Mine development, South Africa	81.9	0.22	50
	56	0.29	50
Delaware Aqueduct	227	0.25	
Hollinger drifts, 8½ by 7 ft.	60	0.218	50
Drifts, various	35–56	0.26–0.63	40–50

NOTE.—Tunnels are in granite or equivalent rocks; South African mine development is in quartzite; drifts are in moderately hard rock.

these figures, a coal-mine adit 8 by 11 ft. required only 0.03 lb. per cu. ft. of 50 per cent dynamite. It was undercut.

A shaft in granite at the Newport mine (Michigan), 15 by 21 ft., burn-cut round, required 0.11 lb. per cu. ft. of 80 per cent gelatin dynamite. The King No. 3 shaft (Canada), 18 by 13 ft., required 0.177 lb. per cu. ft. of 40 per cent dynamite in feldspathic rock and 0.105 lb. per cu. ft. in serpentinized peritotite. In medium-hard rocks, 144 to 184 sq. ft. shaft sections required from 0.22 to 0.35 lb. per cu. ft. of 40 per cent gelatin; raises of 50 sq. ft. section required 0.33 lb. per cu. ft. of 40 per cent dynamite; a 24 sq. ft. section, 0.4 lb. per cu. ft. of 40 per cent dynamite. In softer rocks about half of these quantities would be required.

No very accurate examples can be given for stoping, as the conditions are so varied. However, Fig. 70 gives a consistent comparison of various kinds of mining work in relation to amounts of explosive used. The explosive was 40 per cent dynamite; the rock was hard. Physical characteristics of rock masses, width of stope, and method of mining influence the explosive ratio. Long ago, H. W. Seamon showed that deeper stope holes break more tonnage per shift, and in modern practice there is a marked tendency to increase the length of stope holes wherever possible. Hardness of rock and narrow stope widths usually increase explosive ratio. Some examples are as follows: Gold mines in a single district gave for 4-ft. stopes 0.18 lb. per cu. ft. for 40 to 55 per cent dynamite; for 8-ft. stopes, 0.11; for 12-ft. stopes, 0.09; and for 15- to 65-ft. stopes, 0.05. At Noranda (Canada) stoping in heavy sulphides with long drifter-drill holes and 60-ft. stopes required 0.034 lb. per cu. ft. of 40 per cent dynamite; and in rhyolite ore bodies, diamond-drill holes in 75-to 100-ft. stopes required 0.015 lb. per cu. ft. for primary breaking and 0.038 for secondary blasting, or a total of 0.053 lb. At Tennessee Copper, sulphide ore bodies required 0.041 lb. for primary and 0.021 for secondary, or a total of 0.062 lb. per cu. ft. At Mt. Isa (Australia) sublevel stoping required 0.074 lb. per cu. ft. of 40 to 60 per cent dynamite.

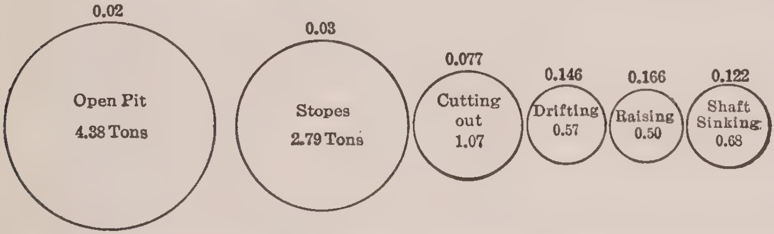


FIG. 70.—Tons broken per pound of explosive. Upper line of figures—pounds of explosive per cubic foot of rock broken.

At United Verde (Arizona), massive sulphides required 0.07 lb. per cu. ft. of 50 per cent dynamite. The Mother Lode mines (California) use about 0.08 lb. per cu. ft. of 30 to 40 per cent dynamite. Alaska Juneau uses 0.004 lb. per cu. ft. for primary and 0.024 for secondary blasting, a total of 0.028 lb. per cu. ft., this low explosive ratio being due to partial caving in wide stopes.

Coal Mining.—Shooting from the solid requires some 0.06 lb. per cu. ft. for black powder; undercut coal 0.01 to 0.013 lb. per cu. ft. of black powder. Shooting from the solid is practiced only where undercutting, shearing, or top cutting is impracticable. Breaking coal is a different problem from breaking ore or rock, as coal is sold as mine-run or is cleaned, sized, and sold in various sizes. Thus a large proportion of lump coal is the objective in blasting.

Salt, Potash, and Phosphate Mining.—Room-and-pillar methods of mining salt indicate a burden of 12 to 20 cu. ft. per foot of drill hole, an explosive ratio of 0.02 to 0.06 lb. per cu. ft. of 25 to 35 per cent dynamite, and an undercut of 7 ft. Room dimensions or sizes of face vary in different mines; 10 by 50 ft. or larger will give an idea of a unit area. Face areas in potash mining in New Mexico are 10 by 20 ft., with an undercut of 8 ft., a burden of 7 cu. ft. per foot of drill hole, and an explosive ratio of 0.06 lb. per cu. ft of 25 per cent dynamite with 43 per cent used in the primer cartridge. Phosphate mining in Idaho required 0.05 lb. per cu. ft. of 20 per cent dynamite; burden on drill hole was about 5 cu. ft. per foot.

Long-hole Stoping.—Summations of practice in long blasthole stoping were made by O. V. Lindqvist. The following figures are taken from one of his tables: Vertical ring drilling, 41 holes aggregating 1,520 ft. broke 2,200 tons, or 1.45 tons (17.4 cu. ft.) per ft., explosives 0.41 lb. per ton. Horizontal parallel holes aggregating 13,865 ft. broke 24,470 tons or 1.77 tons (21.2 cu. ft.) per ft., explosives 0.38 lb. per ton. At Noranda 153 holes aggregating 5,842 ft. broke 22,538 tons or 3.86 tons per ft., with explosives 0.18 lb. per ton.

Sublevel stoping practice was greatly stimulated by Noranda's successful application of long holes in breaking. The light diamond drill and the low-cost bort bits facilitated the drilling of deep blastholes and lowered their cost, thus giving still further impetus to use of such holes in breaking. Diamond-drill holes, horizontal, vertical, or ring-fanned, are now considered applicable not only in sublevel stoping but also in shrinkage stoping and pillar mining.

Stoping Costs.—Breaking costs in stopes are influenced to some extent by the mining methods employed and by local conditions, especially by the amount of timbering necessary. Most stoping costs are inclusive of items that preclude the segregation of the breaking cost.

The following figures illustrative of mining methods at United Verde are taken from a paper by J. B. Pullen: Remnant stopes 10.46 tons per man-shift, 0.44 lb. explosive per ton, 7.47 board ft. timber per ton; vertical pillar stopes 9.06 tons per man-shift, 0.35 lb. explosive per ton, 5.89 board ft. timber per ton; Mitchell-slice stopes 9.35 tons per man-shift, 0.52 lb. explosive per ton, 7.18 board ft. timber per ton; inclined cut-and-fill stopes 11.87 tons per man-shift, 0.78 lb. explosive per ton, and 4.08 board ft. timber per ton. Only a fractional part of the labor is required for drilling, charging, and firing; the remainder is for moving the ore, filling, and timbering.

Estimating Blasting Charges.—First consideration is given to the hardness and other physical characteristics of the average type of rock, ore, or mineral. A silicified rock is more difficult to blast than an unaltered rock; an altered rock is more easily broken than an unaltered one; a rock mass divided by joint, sheeting, or stratification planes is far weaker than one showing no structural discontinuities. The dimensions of the face and probable depth of the round are next considered; its cubic content is then estimated, the explosive ratio and the kind of explosive are selected, and the weight is calculated. The pattern of the round is designed with reference to drilling equipment and drill supports. The estimated weight of explosive divided by the number of drill holes will give the average charge per drill hole; cut holes and lifters are usually given larger charges than intermediate and peripheral holes. The grade of explosive, the cartridge size, whether a fuse or an electric detonator is to be used, and other blasting details are considered. In open-pit bench blasting and tight-face blasting, underground operators have a great store of experience upon which to draw; most manufacturers of explosives have experienced men who advise upon the use of explosives in mining, and most operators who have perfected an efficient use of explosives in their mining have contributed their experience to mining literature.

Ballistic tests are used for grading explosives on a percentage basis or rating by weight based on the standard of 100 per cent blasting gelatin. The ballistic effects of equal weights of the standard and the explosive are compared, the ratio being expressed in per cent. A 25 per cent dynamite is one that shows one-fourth the ballistic effect of a corresponding weight of blasting gelatin; a 50 per cent dynamite shows one-half the ballistic effect, and so on. The speed of detonation, which is as important as the grade, is supplied by the manufacturer for each grade of explosive. Detonating rates for various grades are given in Table 19, page 127.

For the strongest and most homogeneous rocks, 60 to 80 per cent gelatin or ammonia gelatin may be used; for the same rock types weakened by alteration or jointing or otherwise, a 40 to 50 per cent gelatin or ammonia gelatin would be selected. For the general run of stoping in moderately strong quartzes and vein material a 40 per cent explosive is widely used; for softer vein material or for many industrail minerals a 20 to 35 per cent grade may suffice. Sometimes better results are obtained by composite charges, such as a primer cartridge of 50 to 60 per cent gelatin or ammonia gelatin and the remainder of the charge 40 per cent. By varying the proportions of two such dynamites, fragmentation may be brought to the point desired. However, the stocking of too many grades of explosive may lead to confusion in their use. Many mines limit their stock of explosive to three grades.

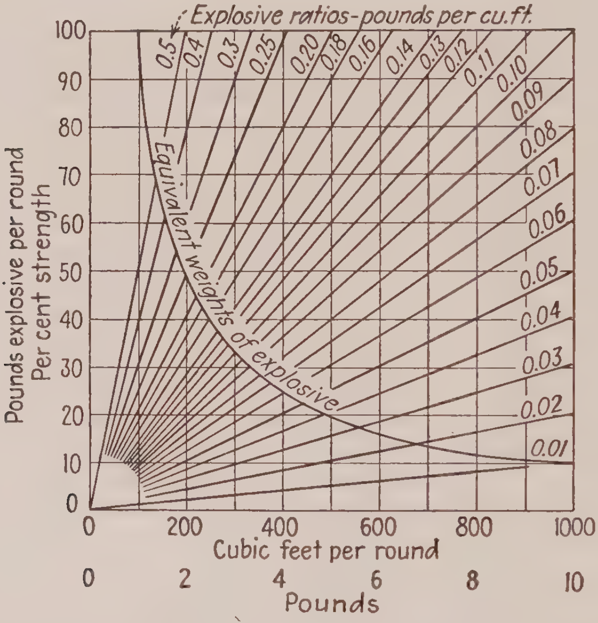


FIG. 71.—Weight of explosive per round at various explosive ratios.

Fragmentation is best achieved by more or less closely spaced drill holes and distribution of charges along much of the length of the hole, as well as by the selection of a grade of explosive and a rate of detonation that will shatter the rock to the desired extent.

Simple charts like Fig. 71 can be constructed to show the weight of explosive or number of cartridges in relation to cubic feet broken in various rounds for shaft, drift, raise, or stope work and for specific explosive ratios. It is sometimes of interest to figure equivalent weights of explosive on the basis of percentage strength relative to the standard. For example, 2½ lb. of 40 per cent dynamite would be equivalent in ballistic effect to 1 lb. of 100 per cent blasting gelatin. In comparing blasting practice it is necessary to compute comparative weights of standard 100 per cent strength, thus reducing various grades to a common basis.

Breaking Size.—There is a limit to which ore or rock pieces may be broken, above which it becomes impracticable to handle them without further reduction in size. In most underground stopes oversize pieces are blockholed and further blasted wherever possible. This also applies to open-pit operations, although there the restrictions with respect to size are usually fewer. In underground work, chutes, car sizes, and loaders impose a limit to the maximum size, which may be 11 to 18 in. in the smallest dimension. Grizzlies are interposed above chutes and transfers to stop oversize pieces and to afford a position for secondary blasting. In some kinds of mining, bulldozing chambers and grizzly drifts may be necessary to facilitate secondary blasting.

Blasting practice is adjusted to avoid excessive oversize breaking as much as possible. In some mines rock breakers are installed underground so as to provide a product that may be more conveniently handled in skips and on conveyors. Except in those cases where ore breaks into small sizes, secondary breaking is always in some degree.

In most open-pit operations dipper sizes establish the size limit, but in many operations careful study is made of blasting practice and sufficient fragmentation is obtained to reduce secondary blasting to very small proportions.

Undercutting, Overcutting, and Shearing.—Coal and minerals of like hardness can be undercut by manual or mechanical methods, thus giving a second free face and an increase in blasting efficiency. Overcutting signifies the placing of the cut at the top of the section. Shearing signifies the making of a vertical cut at the sides of the section or at its mid-point. A face undercut and sheared at both ends presents four free faces and requires a much smaller weight of explosive for breaking than does the solid face. In some cases a horizontal cut is made at a parting at a mid-point of the face, thus assisting in cleaning the coal and giving better conditions for blasting.

Manual methods of undercutting have become obsolete, as has the compressed-air-driven pick machine and post puncher. The greatest use of modern undercutters is in coal mining. In 1938 the bituminous mining industry of the United States employed 1,145 track-mounted cutters, others 967, permissible 3,666, others 6,032, or a total of 11,810; the amount of coal produced annually per machine was 23,566 tons, the machine-cut product being 87.5 per cent of the total coal product. Likewise there were in use 5,071 electric drills and 1,465 pneumatic drills, or a total of 6,536. Electrical operation predominates. In the industrial mineral field, salt, potash, phosphate, and clay beds are undercut as a preliminary to blasting wherever conditions permit.

The undercutting machine (Fig. 72) consists of a cutter head; a chain carrying the cutters and operated by a sprocket drive, gears, and electric motor; a structural frame; and a pair of drums, one on each side, for the feed cables. Roof jacks provide anchorages for the feed cables. Either d-c or a-c motors are used. Undercutting machines are manufactured and designed for several specialized services, as follows: The short-wall cutter used in rooms and entries; the longwall cutter, used in longwall mining; the top cutter and the track-mounted universal cutter, which can be used for bottom, top, center, and side cuts and angle shears. Short-wall cutters have the cutting bar in line with the machine; longwall cutters have the cutting bar at right angles to the frame of the machine. Figure 73 gives details of an adjustable top cutter. Cutters other than the track-mounted type slide or skid along the bottom. The maximum length of cutter bars is 9 ft. and the minimum 2½ ft.; the thickness of the kerf or cut ranges from 2½ to 6 in. Short-wall cutters weigh from 6,500 to 7,600 lb. and are powered with 50-hp. motors; the smallest cutter weighs 1,600 lb. and is powered with a 10-hp. motor; medium-weight machines weigh from 2,200 to 2,300 lb. and are powered with a 20-hp. motor. The track-mounted universal weighs 13 tons; its maximum cutting width with a 9-ft. bar is 40.75 ft.; cutting height above rail is 8.3 ft.; vertical shearing range above track is 10 ft. Cutter bars from 6 to 9 ft. can be equipped to cut 3-, 4-, and 5-in. kerfs; a 7-ft. cutter bar can be equipped to cut a 2¼-in. kerf. Sumping speed is 0 to 8 ft. per min.;

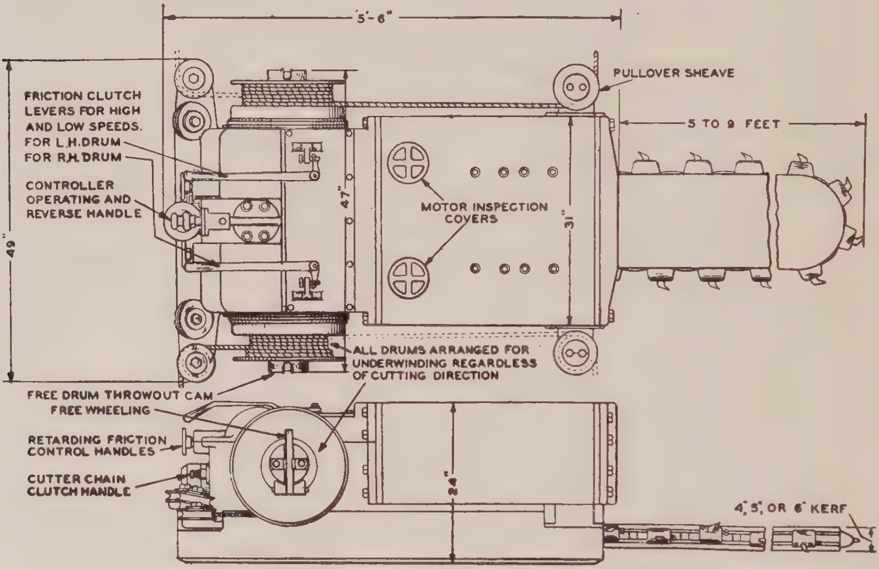

5'-6"
PULLOVER SHEAVE
FRICTION CLUTCH LEVERS FOR HIGH AND LOW SPEEDS.
FOR L.H.DRUM
FOR R.H.DRUM
5 TO 9 FEET
CONTROLLER OPERATING AND REVERSE HANDLE
47"
MOTOR INSPECTION COVERS
31"
49"
ALL DRUMS ARRANGED FOR UNDERWINDING REGARDLESS OF CUTTING DIRECTION
FREE DRUM THROWOUT CAM
FREE WHEELING
RETARDING FRICTION CONTROL HANDLES
CUTTER CHAIN CLUTCH HANDLE
24"
4, 5, OR 6" KERF

Fig. 72.—Short-wall coal cutter. (*Sullivan Machinery Co.*)

swing speed 0 to 50 ft. per min., and travel 0 to 4 m.p.h. Heavy shortwall cutters have variable cutting speeds of 18, 20, 26, 33, and 44 in. per min. and a hauling speed of 34 ft. per min.

Cutters are served by a power cable which is handled by a self-propelled power truck. Unmounted machines are transported from place to place by means of a truck of drop-front construction. Small machines have a bottom-cutting speed of 14 in. per min. and when inverted a top-cutting speed of 7 in. per min.

An undercutting cycle begins with the placing of the roof jack and the dragging of the machine from its truck to a position for the sump cut.

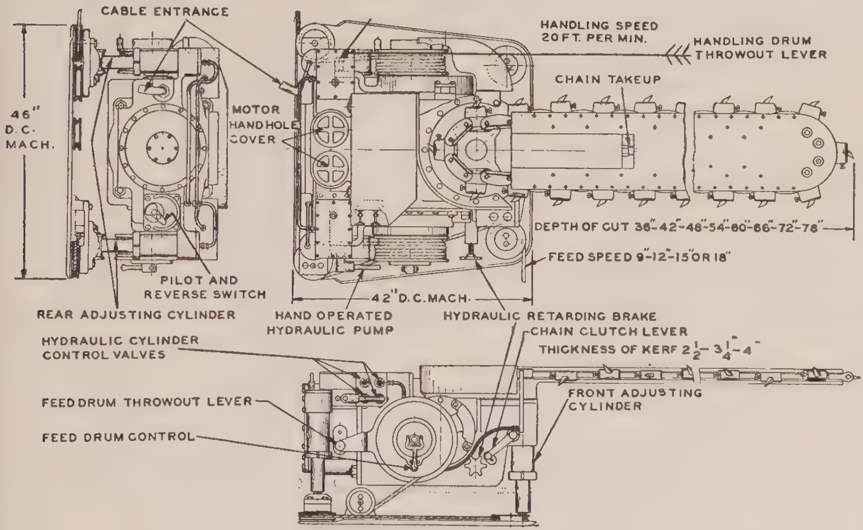

CABLE ENTRANCE
HANDLING SPEED 20 FT. PER MIN.
HANDLING DRUM THROWOUT LEVER
CHAIN TAKEUP
46" D.C. MACH.
MOTOR HAND HOLE COVER
DEPTH OF CUT 36"-42"-48"-54"-60"-66"-72"-78"
FEED SPEED 9"-12"-15" OR 18"
PILOT AND REVERSE SWITCH
42" D.C. MACH.
REAR ADJUSTING CYLINDER
HAND OPERATED HYDRAULIC PUMP
HYDRAULIC RETARDING BRAKE
CHAIN CLUTCH LEVER
THICKNESS OF KERF 2½-3¼-4"
HYDRAULIC CYLINDER CONTROL VALVES
FEED DRUM THROWOUT LEVER
FRONT ADJUSTING CYLINDER
FEED DRUM CONTROL

Fig. 73.—Over coal cutter. (*Sullivan Machinery Co.*)

The roof jack is then moved and placed in position for the sumping cut, which is started at an angle to the face, the cutter being swung into a position normal to the face as it deepens the cut to its full depth. Next, the roof jack is moved and the cutter makes its running cut along the face of the room (Fig. 74). Finally the jack is placed so as to draw the cutter into loading position.

The cycle for a universal consists of moving the cutter into the room, swinging the bar into sump-cutting position, cutting the sump, and continuing the cut as an arc to the other side of the room. As an example of speed, a sump cut of 9 ft. and a swing across a 16-ft. face and out require 4 min.; shearing cuts of about 5 ft. require 3 min. This information was taken from the Sullivan catalogue.

B. Y. Williams states that only 20 per cent of shift time is used in room cutting, the remainder being required for transferring equipment from room to room, unloading and loading, and setting roof jacks. In concentrated working, however, 33 per cent cutting time can be obtained. He gives an example of 400 ft. with a 6½-ft. depth of undercut per 8-hr. shift in longwall coal mining. Power requirements are approximately 44 watt-hours per square foot of undercut. On the basis of coal undercut, power per ton would vary over a wide range, being dependent upon the thickness of the seam undercut. Dust produced in undercutting is allayed by a stream of water delivered upon the cutting chain during the operation. Small cutters are equipped with a water tank and pump for this purpose.

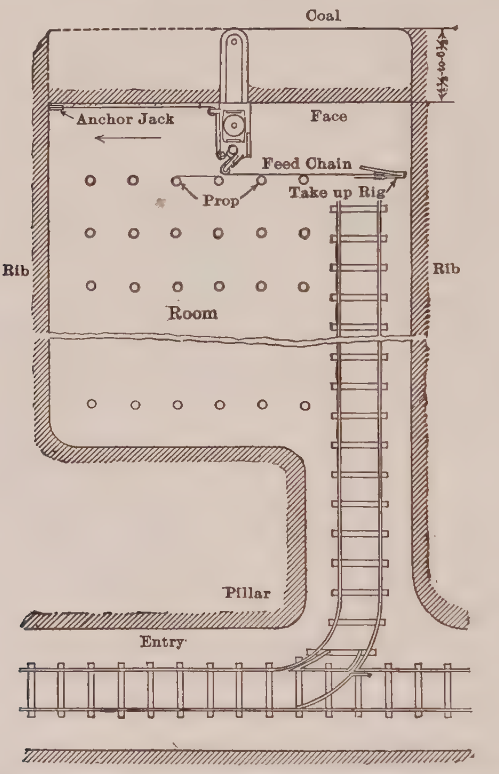


FIG. 74.—Room-and-pillar mining with continuous coal cutter. (*Sullivan Machinery Co.*)

In potash mining, undercutting is at the rate of 6 in. per min., with 9-ft. cutters and 8- to 8½-ft. undercut. Short-wall cutters are used for this purpose. In salt mining 6- and 9-ft. cutters are used; the cutting rate is 6 in. per min. At the Retsof salt mines, 120 to 150 ft. (960 to 1,200 sq. ft.) per shift is undercut.

Miscellaneous Methods.—Tunneling machines and other mechanical excavators have been used but are now of historical interest only. Miscellaneous hand tools are in use to a limited extent. The pick, bar, gad, moil, and sledge have been superseded to a considerable extent by pneumatic tools. The pneumatic pick finds application in shaft sinking for cleaning up the shaft bottom before mucking is completed; pneumatic mauls are used in place of hand sledges for reducing oversize pieces of ore.

The pointed bar finds application in dislodging loose rock and scaling down working places after blasting. It is of use in all stoping operations, bench mining, glory-hole operations, and in development and exploration to test for loose rock and to make working places safe. A light alloy steel-shod rod is also in use.

Breaking by caving methods has assumed great importance in the mining of massive ore bodies. Copper, iron, and molybdenum ores, asbestos, and limestone are won by this method in a number of mines. Preparation for caving involves the driving of narrow work by the usual methods of air drill, blasting, and scraper work. The undercut is drilled and blasted and gravity induces caving with accompanying rock breakage.

The breaker pad, a fabric-reinforced rubber bag, promises important application in coal mining. Hydraulic pressure up to 500 lb. per sq. in. gives an over-all thrust of 75 tons. The pad is inserted into sawed slots, and the coal is brought down by pressure.

Not much information is available about the results obtained by the use of breaking pads. In the preceding chapter the du Pont expansible tube was mentioned as a substitute for explosives in coal mining. C. H. Lambur, Jr. (*Coal Age*, April, 1940), gives some results of this method of breaking in mining coal. A 7-ft. tube, consisting of a 1-in.-wall rubber tube enclosed in a braided mesh cover of high-tensile steel wire with suitable closures and connections for high-pressure hose, is expanded under hydraulic pressure from 4.75-in. up to 7.5-in. diameter or more. The pressure may reach 2,000 lb. per sq. in. The hydraulic medium is a special oil of low inflammability supplied by a self-propelled pumping unit. The tube is used in a 5½-in. diameter borehole. The coal face must be undercut, and shearing would be of advantage. Lambur states that a hydraulic unit at the New Monarch mine, Herrin, Ill., broke down sufficient coal for two loading machines, each averaging 300 to 320 tons per shift. This required breaking 10 faces in a 7-hr. shift. The war appears to have stopped further experimentation with these methods of breaking.

Cost of Breaking.—The principal cost items in primary breaking are labor, power, explosives, supplies, drill sharpening, drill steel, supervision, maintenance and repair of drills and machines, depreciation of mechanical equipment, and a proper proportion of overhead expense. Secondary breaking involves much the same items but on a much smaller scale. In the case of undercutting there is some reduction in the amount of drilling and explosives required, but there is the cost of undercutting, which includes labor, power, supplies, and other items.

Open-pit breaking is lowest in cost; block-caving and caving are comparable in cost to open-pit but are usually higher; in ascending scale are large stopes, large rooms, large underground excavations such as hoist stations and pump rooms and greatest in cost are all narrow workings such as exploratory and development openings of various kinds. Hard rock costs more to break than softer kinds. Other things being equal, the greater the face area the more effective is the breaking and the lower its cost; an increase in bench height in open-pit mining may be expected to give lower costs. In any given case research may result in the discovery of more advantageous adjustments in methods of drilling and the use of explosives. Figure 75 indicates the relative costs of breaking, but present-day costs are almost double those indicated.

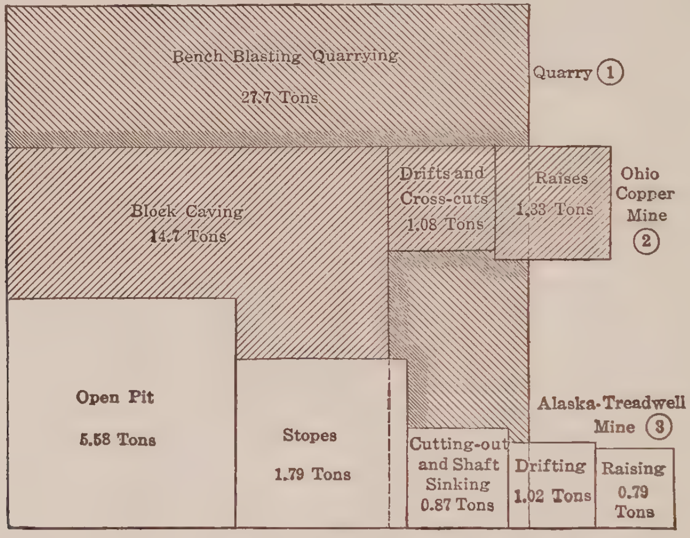


FIG. 75.—Comparative costs of breaking—tons per dollar; present day costs are from 1.5 to 2 times above.

Another comparison of recent date of the influence of dimensions and method is afforded by some of the performance records at Copper Mountain, British Columbia (*T.P.* 1914, *A.I.M.E*). The figures are given in the accompanying table.

COMPARISON OF MINING METHODS AT COPPER MOUNTAIN

Details	Glory holes	Shrinkage stopes	Open stopes	Large blasts
Tons per drilling shift	195.4	86.9	29.7	300.2
Tons per miner shift	130.3	84.5	29.1	218.2
Explosives, lb. per ton	0.172	0.379	1.302	0.176

Loading and transportation, vertical, horizontal, or both, are accessories to rock breaking, so that costs frequently include these items. The breaking problem is closely related to these succeeding operations, and all three must be coordinated for satisfactory results. These subjects are considered in the succeeding chapter.

The following example of the cost of bench breaking is taken from Blasting Practice at New Cornelia (Ajo, Ariz.), by H. A. Angst and R. A. Cochrane.

Type of material	Monzonite, hard	Tons broken per ft. drilled	43.9
Number of holes	51	Loading calculated, lb. per cu. yd	0.7
Average height of bench	59	Bottom charge, lb	22,450
Maximum depth of holes	67	Deck load, lb	6,850
Total drill footage	3,417	Cost of drilling	$3,229.28
Tons broken	150,000	Cost of blasting	3,092.23
Pounds explosive	29,300	Total cost	$6,321.51
Tons per pound	5.1	Cost per ton broken	0.0423
Hole spacing, feet	15	Primacord used	4,500 ft.
Distance from crest, ft.	12	Explosives per ton, lb	0.190
Distance from toe, av. ft.	19.5		

At the New Cornelia mine in 1939 the breaking of over 14,000,000 tons of hard and siliceous waste- and ore-rock formations gave 5.48 tons broken per pound of explosives; 0.4 ft. of Primacord and 0.08 ft. of fire fuse were required per pound of explosive; blasting caps approximated one per 28.7 lb. of explosive; total explosives consumed 2,554,000 lb. C. E. Mills gives the cost per cubic yard of coyote-hole blasting at the United Verde open pit as development $0.083, explosives $0.082, labor, supplies, and miscellaneous $0.011; total $0.176 per cubic yard; average 1.151 lb. explosive per cubic yard, principally black powder.

Notable large blasts have been made in both metal mines and quarries. At Climax (Colorado) an area 150 by 425 ft. by 250 ft. in height was undercut and blasted by means of coyote holes in which 14.9 tons of 50 per cent, 13.1 tons of 60 per cent, and 27.1 tons of Gelex No. 2 were distributed. Expectation was 850,000 tons and primary breaking 8 tons per pound of explosive. At Anyox (British Columbia) a 500,000-ton blast underground gave the following costs: 5,595 long holes $9,890; powder drifts, 398 ft., $2,718.34; loading, back filling, explosives, and hookup $18,104; a total of $30,712.34 or 6.13 cents per ton.

Reports of development costs are usually difficult to analyze on the basis of breaking cost, but the following approximations may be of use: For hard-rock drifts 5 by 7, 20 cents per cu. ft.; for soft rock, 10 cents per cu. ft. At the Homestake mine 7- by 7-ft. drifts in hard rock cost 0.18 cents per cu. ft.; pilot raises 6 by 6 ft., 38 cents per cu. ft. Stripping a 19.25- by 14-ft. shaft to pilot raise cost 8 cents per cu. ft.; the shaft section cost 12.3 cents per cu. ft. At the south main Flin Flon shaft, the sinking of a 14- by 20-ft. shaft cost 12.2 cents per cu. ft.; a pilot winze 7 by 10 ft. cost 30.6 cents per cu. ft.; a pilot raise 5 by 5 ft. cost 73 cents per cu. ft., and enlarging the shaft section cost 10 cents per cu. ft.

Breaking costs in stoping vary with the hardness of the rock, the width of the stope, and drill-hole depth. Narrow stopes, 2 to 4 ft. in width, may cost from 20 to 25 cents per cu. ft.; stopes 4 to 10 ft., 10 to 15 cents per cu. ft.; wide stopes, 5 to 10 cents per cu. ft. Only breaking is included in the foregoing figures.

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CHAPTER VII

TRANSPORTATION AND HOISTING

TRANSPORTATION

The transportation of ore, waste, timber, men, and supplies on the surface and underground is one of the important problems arising in mining. The problem involves loading, unloading, and transference between two or more points. The points may be in the same horizontal plane, one above the other as in a shaft, or there may be more or less difference in elevation between the points, either in favor of the load or against it. The problem is complicated by the interspersing of level parts with grades and the necessity of using several different methods of transportation in bringing the ore from the face to the shipping point.

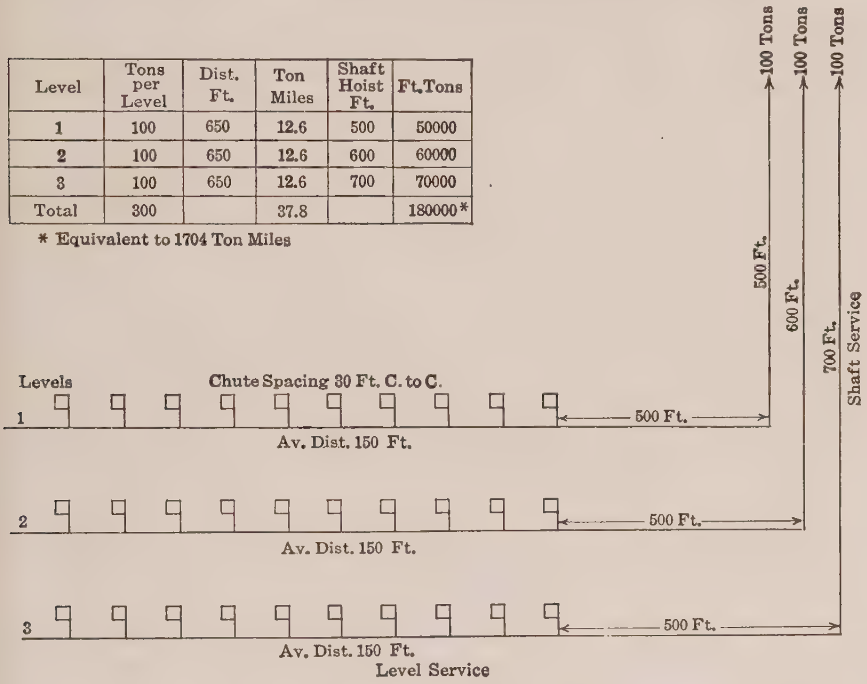
Underground Transportation.—In most metal mines several transfers are required. From the stope face the ore is shoveled into chutes, from which it is drawn into cars on a given level and trammed to the shaft where either the car is loaded upon a cage or the ore is dumped into a shaft pocket, from which it is loaded into a skip and hoisted to the surface. In either case it is dumped into a bin from which it is transferred to a mill or stock pile or loaded into cars, trucks, or other vehicles to be transported to its destination. Each level of a mine may be equipped with a track, cars, and chutes. Hand tramming or mechanical haulage may be employed. In some mines the ore is dropped through inclined transfers to a main haulage level upon which ore trains are operated.

In coal mines the coal is loaded at working places into cars by hand, mechanical loaders, or conveyors and then hauled to gathering points, where the cars are made up into trains. The trains are hauled to the surface by locomotives, or they may be hoisted up an incline. The cars are discharged at a tipple, from which the coal is conveyed to a preparation plant or cleaning plant. The prepared product is loaded into railroad cars for transportation to market.

In both types of mines supplies are handled on return trips, and waste rock that cannot be utilized underground is hauled in cars and by-passed at the surface bins into waste-rock bins from which it is transferred to waste-rock dumps.

In Fig. 76 the haulage service for a metal mine producing 300 tons per day is illustrated. The loading points, tons, and distances are indicated in the chart. Three levels are assumed to be producing ore, and

10 loading points are represented upon each level. The ton-miles or service required upon the levels is in the aggregate 37.8, and the shaft service 180,000 ft.-tons. The latter is equivalent to 1,704 ton-miles, the figure being obtained by assuming that 1 ft. of vertical distance is equivalent to 50 ft. of horizontal distance. The relative importance of level haulage and shaft service is brought out by the comparative figures. In most metal mines of small tonnage the ore transportation upon the levels is of small importance, so that simple methods such as hand tramming or horse haulage are more economical than methods requiring a more elaborate equipment.



Level	Tons per Level	Dist. Ft.	Ton Miles	Shaft Hoist Ft.	Ft. Tons
1	100	650	12.6	500	50000
2	100	650	12.6	600	60000
3	100	650	12.6	700	70000
Total	300		37.8		180000*

* Equivalent to 1704 Ton Miles

FIG. 76.—Transportation in a metal mine.

Coal mines differ from metal mines in that they usually have only one working level and the mine cars can be brought to the face and loaded. The cars are loaded by shoveling. The horizontal distances are much greater than those common in metal mines. In handling the coal, transfers from car to shaft pocket are avoided in order to reduce breakage to a minimum. This necessitates cage hoisting. All the cars must be taken from the faces, delivered to the shaft bottom, loaded upon the cages, hoisted, unloaded, and sent to the tipple where they are discharged. The tonnages handled are usually much in excess of metal-mine tonnages and range from 500 to 5,000 tons per day.

Haulage in coal mines is divided into "gathering service," which comprises the removal of the cars from the working faces to a parting at or close to the point at which the side entries begin, and "main haulage service." In main haulage the trains of cars that are made up at the parting are hauled either directly to the tipple or else to the shaft bottom.

Entry	Tons per Room	Total Tons	Av. Dist. Feet	Av. Dist. Miles	Ton Miles
1	30	300	3000	0.57	171
2	30	300	3500	0.66	198
3	30	300	4000	0.76	228
4	30	300	4500	0.85	255
5	30	300	5000	0.95	285
Total		1500			1137

For Hoisting 1500 Tons-500 Ft.-Service Equivalent to 7080 Ton-Miles

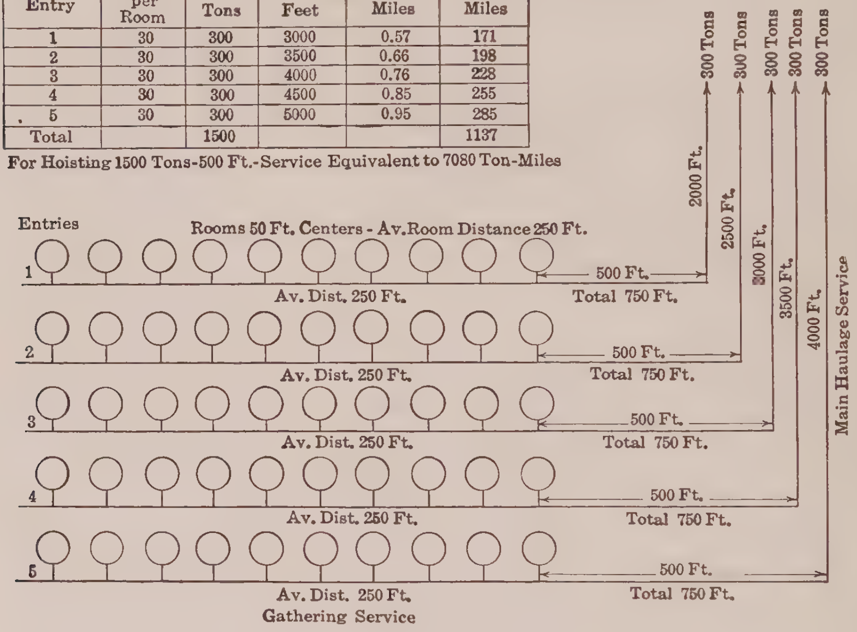


FIG. 77.—Transportation in a coal mine.

The system usually met with involves gathering, main haulage service, and shaft service. Horse haulage or light locomotives are used for gathering service and locomotives for main haulage.

In Fig. 77 the haulage service of a coal mine producing 1,500 tons per day is illustrated. The working places, distances, and tonnages are given. The aggregate ton-miles of service per day is 1,137. The importance of the haulage problem in a coal mine needs no further comment.

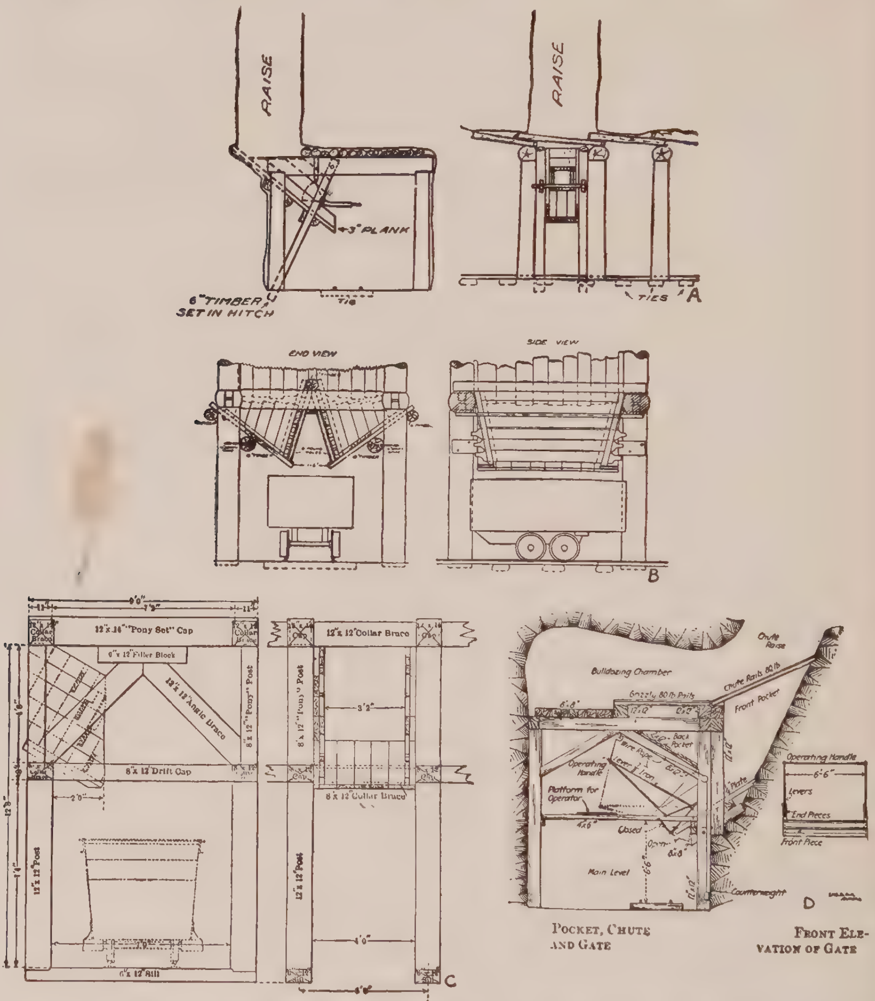
The methods of underground ore movement are summarized as follows:

NONMECHANICAL:	REMARKS
Chutes, vertical transfers, and troughs inclined at such an angle as to produce gravity flow	Limited to relatively short distances at angles ranging from 35 deg. to vertical in favor of ore movement
Wheelbarrow, car and track, hand tramming, horse and mule haulage	Limited to low grades and relatively short horizontal distances

MECHANICAL:	REMARKS
Scraper	Limited to horizontal range and to angles below which gravity flow ceases
Conveyors—belt, chain, and shaking pan	Limited to short horizontal distances and to moderate angles
Shuttle cars	Limited to comparatively short distances and low angles
Rope haulage and gravity planes	Limited to low angles, but in the case of the gravity planes high angles may prevail; distances usually short
Locomotive haulage	Limited to grades of 2 to 4 per cent against the load, but distances may be very great relative to those of other methods

Loading.—Cars or conveyors are loaded by hand shoveling, by chutes, conveyors, mechanical loaders, and shuttle cars. Development headings require greatest man power in shovel loading, least in chute loading. More or less hand shoveling will always accompany mining operations; but scrapers, conveyors, and mechanical loaders have relegated much hand shoveling to small initial exploratory and prospecting work. Chute loading and hand tramming of cars are old-time methods that still find a place in many metal mines and will continue to be important. Chute loading is usually the lowest in cost and is employed wherever possible. Where conveyors are used, as in some methods of coal mining, the conveyor discharges into an elevating conveyor to obtain car height, or the discharge end of the conveyor is raised sufficiently to command car height.

Chute Design.—Many designs of loading chutes are exemplified in mining practice (see Figs. 78, 79, and 80). For hand tramming, chute widths are 2 to 2½ ft. and lip discharge above car height is 4 to 6 in. Controls range from several boards held by cleats to metal overcut arc gates. Cars of 20 cu. ft. capacity or less are loaded. Larger cars require greater chute widths to ensure rapid loading, the width being approximately one-half the car length but limited to a maximum of 6 ft., clearance above the car being proportionately increased. Cars of 40 to 200 cu. ft. capacity are loaded from chutes 3 to 6 ft. in width. The height of the chute opening is three-quarters or more at the width. Large chutes require quick-acting gates, preferably operated from a platform above car level and in front of the chute. Compressed-air cylinders are usually employed for operating the gates. Gates are overcut arc, undercut arc, undercut sliding, and hinged apron stops. For coarse ore, various designs of the finger-chute gate (wooden or metal bars or chains and attached balls) give excellent control. In many designs of large chutes an upper or check gate is also included. Where fines are in force, the gate should be designed to prevent spillage. Slopes of chute bottoms are 35 deg. Chute spacing ranges from a minimum of 25 ft. to 50 ft. up to various distances determined by the method of mining and the supplemental ore movement by scrappers to transfers, which transfers become the loading points. In large stopes where many chutes may be required the spacing is some multiple of the car length, so that loading may take place simultaneously at several points along the train length.



Pocket, Chute and Gate

Front Elevation of Gate

Fig. 78.—Types of ore-loading chutes. (*A*) and (*B*) Mesabi Range (*Univ. Minn.*); (*C*) Ray mine (*Trans. A.I.M.E.*); (*D*) Bulldozing and loading chute (*Eng. Min. Jour.*)

Shoveling.—Shoveling and hand tramming are sequential operations usually performed by the same crew. Shoveling from a rock bottom is

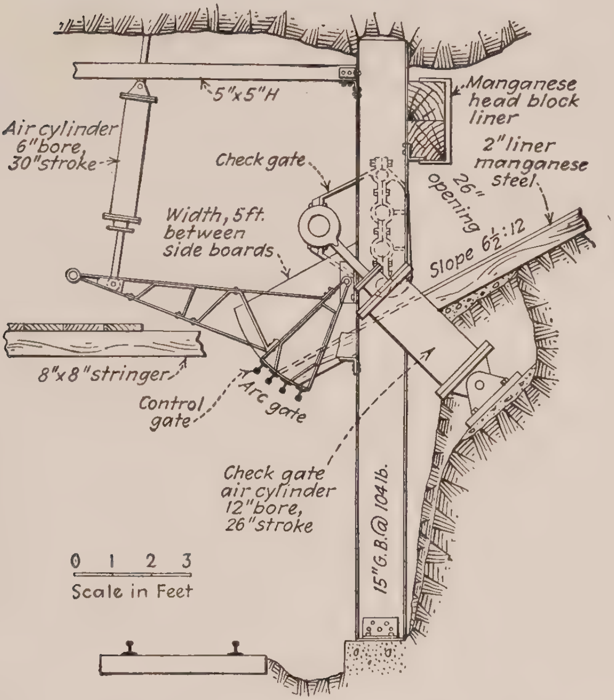


Fig. 79.—Heavy chute gate, air-operated. (*Alaska Juneau mine.*)

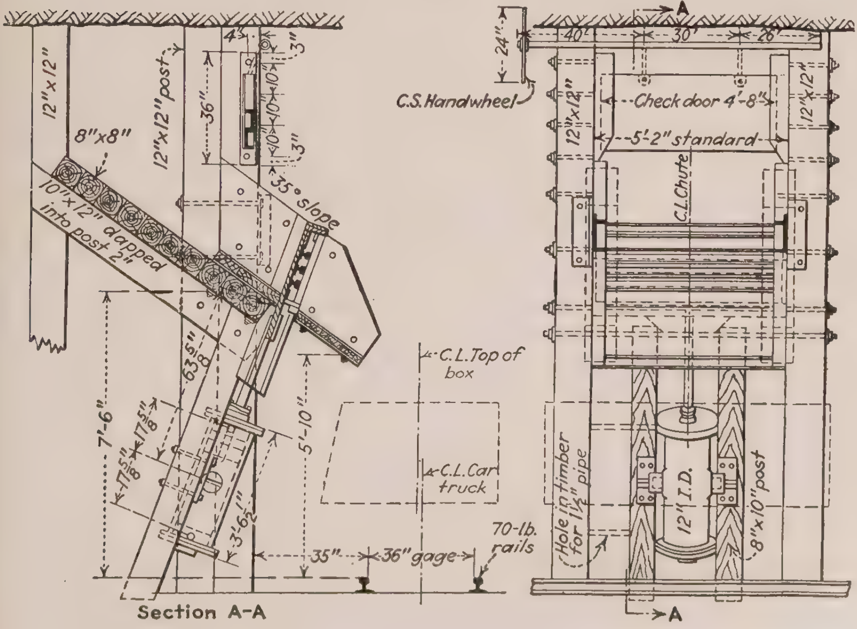


Fig. 80.—Air-operated chute gate. (*Sullivan mine, Kimberley, B. C.*)

the worst arrangement; the best is from a board plat or iron sheet. For tramming distances of 300 to 700 ft. or more, the amount moved ranges from 1.52 to 1.79 tons per man-hour; for distances of 1,000 ft. the amount is 1.17 to 1.6 tons. In shoveling from a plat, the range is from 1.75 to 2.1 tons per man-hour for distances up to 450 ft.; for 1,000 ft. a rate of 1.75 tons per man-hour was given in one instance. For chute loading and tramming, rates were 10.3 tons for 150 ft., 4.21 tons per man-hour for 300 ft.; the average for 11 Michigan copper mines was 14.5 tons per man-shift over an average distance of 609 ft. G. Townsend Harley gives much detailed information about shoveling in *Transactions* of the A.I.M.E., vol. 61, p. 147.

Scrapers.—The two-drum and three-drum hoists used for scraper work are available in numerous sizes, compressed-air or electrically operated, and ranging from 10 to 150 hp. Two speeds are sometimes provided, one for digging and a higher speed for dragging after the scraper has taken its load. Rope speeds range from 130 to 280 ft. per min. but on heavier installations have been increased to a maximum of 450 ft. per min. (pulling load) to 650 ft. per min. for the empty scraper (C. F. Jackson and J. H. Hedges). Power requirements range from 0.065 to 0.5 kw.-hr. per ton for long, difficult drags.

Most scraper designs are of the hoe type, with widths of 30, 36, 42, 48, and up to 60 in. In the best designs the blade is curved and is provided with a replaceable edge; the short and long harness gives over-all lengths of 52 and 72 in., respectively, and both are designed to give balance. Types are open hoe and a hoe with side plates, 18, 24, and up to 40 in. in length and of such depth as to leave the lower third of the blade exposed. Open-blade types are used on short hauls where there is a groove; side-plate scrapers are best in medium and long hauls and for slide loaders. Weight and balance are important. Open types weigh from 467 lb. (6 cu. ft.) to 676 lb. (13 cu. ft.); the heaviest, long-harness types with 24-in. side plates, 36 cu. ft. capacity, weigh 1,402 lb. and require a 50-hp. unit for operation. Often additional weights are added to secure better balance and to get greater digging power.

The scraper loader consists of a track-mounted slide, a tread-tractor slide, or a movable timber slide upon which the power unit is mounted and which delivers at car height. In each case the size of the scraper and car determines the design.

The capacity of scrapers depends on the type of service, the distance involved, the size of scraper, the bottom over which the material is to be moved, the rope speed, whether the material is coarse or fine, and the number of moves or adjustments required at a given working place. Greatest capacity is obtained in scraper drifts where ore is moved a maximum distance of 150 ft. with heavy scraper equipment to chute openings in amounts up to 500 tons per shift or more. A. J. McDermid gives some Miami scraper-loader rates in drifts as 15 to 25 cars loaded in 2 hr. and moved 500 ft. to the switch. Four scraper loads, filling a car, required 3 min. C. W. Nicholson gives the following rates for flat stopes in the Tri-State district: Scraping to a hopper, 144 tons per shift, 48 tons per man-shift; scraping with permanent ramp into cans, 190 tons per shift, 95 tons per man-shift; scraping with portable table ramp into cans, 182 tons per shift, 61 tons per man-shift; scraping with two semi-portable ramps into 2.5-ton cars, 123 tons per shift, 61 tons per man-shift; scraping to conveyor belts, 200 tons per shift, 100 tons per man-shift.

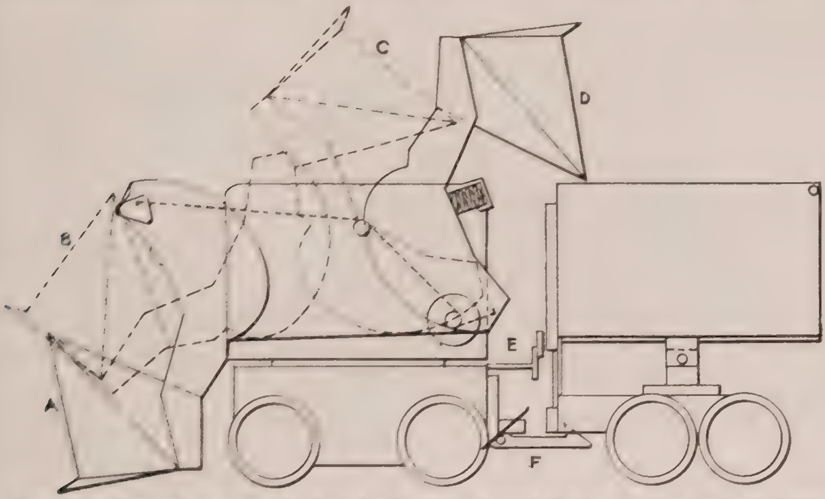


Fig. 81.—Vertical arc-type loader. A—bucket at start of thrust; B—bucket loaded at end of thrust; C—bucket when caging loader; D—bucket in dumping position; E—car bumper; F—car coupler. *(H. M. Courtney, E.M.J.)*

Mechanical Loaders.—Most large-scale operations in metal, coal, and industrial minerals employ mechanical loaders. No one loader meets all mining conditions; hence loaders are more or less specialized. Two principal types are the mobile and the track-mounted. The mobile loader, a tread-tractor-mounted machine, is independent of tracks. The power shovel, used in salt, limestone, and open-stope lead and zinc mines, is a smaller and more specialized design than those used at the surface. Dippers range from ½-, ¾-, and up to 1¼-cu. yd. capacity and electric power is almost universally used. The St. Joseph shovel is a low type, full revolving, weighing 13 tons; it has a 32-ft. sweep and a special dipper that discharges over a rear lip by turning upon side pivots. It was developed for the open flat stopes characteristic of some lead mines. The Keystone and Nordberg (Butler) loaders are also of special design for underground service. The Joy mobile loader was designed for mechanical loading in coal mines but is also used in salt, potash, and limestone mining. It consists of two digging elements on either side of a flight conveyor that discharges into the mine car; its loading boom can be swung through an arc of 90 deg. It is used for car, conveyor, and shuttle-car loading.

Track-mounted loaders are the Conway, Eimco-Finlay, Sullivan, Gardner-Denver, and others of the dipper type, and the Goodman, Meyers-Whaley, Jeffrey, and similar types comprising digging elements in combination with flight or chain conveyors. The dipper-type loaders are designed for rock and ore and the others for coal and less abrasive materials.

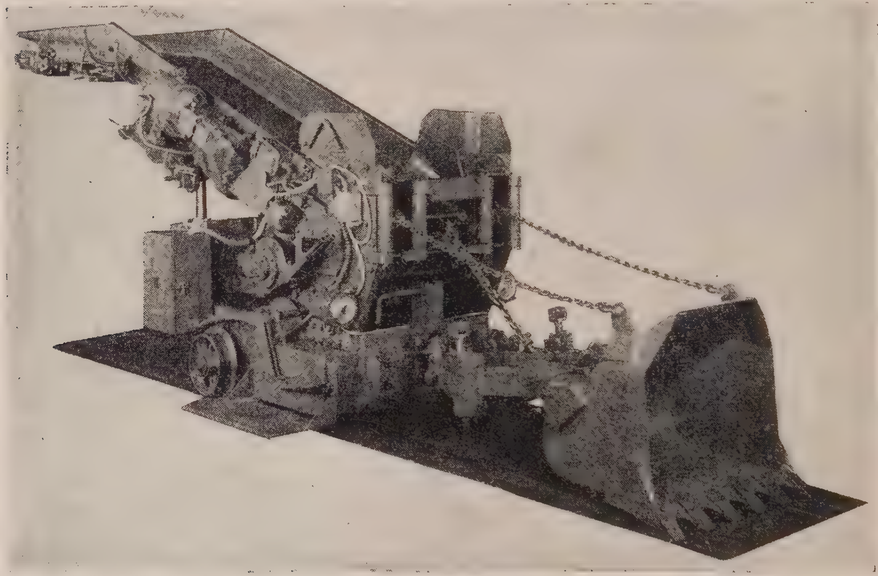


Fig. 82.—Conway loader. (*Goodman Manufacturing Co.*)

Vertical arc-type dipper loaders (Fig. 81) are extensively used in metal and industrial mineral mining. They are operated by compressed-air engines. The Conway, a combination bucket and conveyor (belt) (Fig. 82), can be operated by a compressed-air engine or by electric motors. The coal loaders are all electrically operated.

Vertical arc-type loaders are designed for drifts 7 to 10 ft. in width; the Conway shovels range from a minimum cleanup width of 6 ft. to a maximum of 20 ft. Top clearances vary with the size of the loader, manufacturers providing a range of sizes to meet varying operating conditions. The coal loaders (Fig. 83) range from conveyor loaders for a 30-in. seam, shuttle-car loaders for a 42-in. seam, and car loaders for seams 48 in. high and upwards. Where cars are to be loaded, loading height and car length are taken into consideration. The abrasive nature of the material to be handled plays an important part in the selection of a loader as well as in the material from which its wearing parts are constructed.

The capacity of loaders depends upon availability of cars and transportation facilities. In driving the Stanislaus tunnel (California), a Conway loader mucked out a round in 87 min., or at the rate of ½ cu. yd. per min. As the number of rounds per day was 4.28, the loader was employed 372 min. per day. In the Carlton drain tunnel, an Eimco-Finlay loader averaged 4 min. per 3-cu. yd. car for loading and switching, or ¾ cu. yd. per min.; mucking time required 11.1 to 18.4 min. per foot of tunnel, the cross section of the tunnel averaging 3.8 to 4.4 cu. yd. per ft., place measure. Both of the foregoing tunnels were driven at maximum speeds. In the Butte district, vertical arc loaders are employed in development, and about half the stopes are silled for loading out by such loaders. H. M. Courtney gives average loading time of 1 min. for ¾-ton cars, 2½ min. for 2-ton cars, and 4 min. for 4-ton cars.

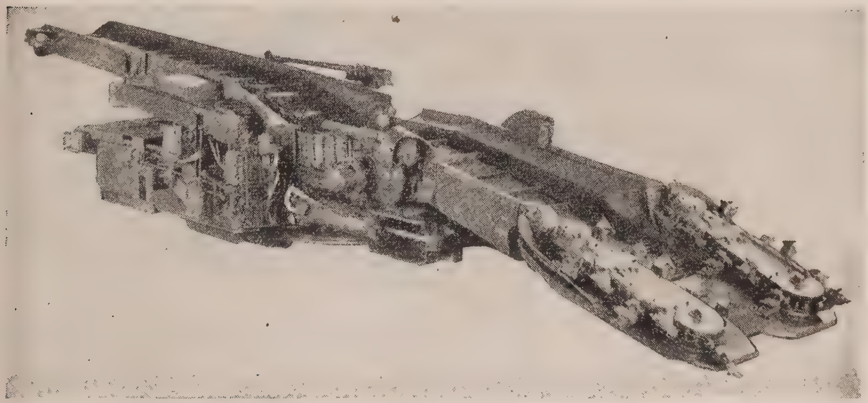


Fig. 83.—Track-mounted coal loader. (*Goodman Manufacturing Co.*)

In potash mining at Carlsbad (New Mexico), a Joy, type 11BU, averaged 83 cars or 433 tons per shift, loading out from large rooms. In a Michigan salt mine, a ¾-yd. dipper shovel loaded 200 cars or 600 tons in a 9-hr. shift. At the Retsof salt mine (New York), a ½-yd. loader averaged 350 tons per shift into 5-ton cars, about 0.5 kw.-hr. per ton loaded being required. Coal loaders average from 2 to 5 tons per min., depending upon the size of the machine. Output per shift is determined by mining conditions and haulage.

Mechanical loading may be suitable in large-tonnage operations, but in small mines of the 100 to 300 tons per day class, mechanical loading is of questionable application except where horizontal development can be concentrated. In tonnage operations, haulage and hoisting operations must be such that mechanical loaders can be kept busy. The largest field for mechanical loaders is in coal mining and in tunneling where time is of importance. Dimensions of mechanical loading machines are given in Table 26.

TABLE 26.—MECHANICAL LOADING MACHINES

Conway loader	20	20 B	50	75	Dimension
Operating height	70	82	131	131	In.
Tramming height	52	67	92	92	In.
Tramming length	24	21	29	29	Ft.
Weight	7.5	8.5	12	15	Tons
Track gage	24	24	36	36	In.
Motor hp.	20	25	50	75	
Loading capacity	22.5	25	45	75	Cu. ft. per min.
Tramming speed	100	150	176	176	Ft. per min.
Minimum radius	12	13	20	20	Ft.
Cleanup width	6	10	15	15	Ft.
Dipper capacity	4.5	5	9	13.5	Cu. ft.

Coal loaders	Goodman 260	Meyers-Whaley No. 3	Joy 7 BU	Joy 14 BU	Dimension
Tramming width	6	5.8	6	6.1	Ft.
Tramming height	54	44	43	34.4	In.
Length	23.3	29	24.75	24.3	Ft.
Weight	13	9.3	8.3	7.8	Tons
Track gage, minimum	36	20	.....	....	In.
Main drive motor	50	25	.....	32.5	Hp.
Cleanup sweep	24	24	.....	....	Ft.
How mounted	Track	Track	Mobile	Mobile	
Capacity	....	3	2	5	Tons per min.

Vertical-arc loaders	Weight, lb.	Over-all width, in.	Over-all length, in.	Digging width, in.	Loading cycle, sec.	Head-room required, in.	Bucket cap'y, cu. ft.
Eimco 12-B	4,000	30	70	76	6	72–83	5½
Eimco 21	6,000	33	81	85	7	87–95	7
Sullivan	4,200	32.75	74–80	84	..	79–87	4
Gardner-Denver	4,300	32	76.25	60–96	..	76	4

Mule Haulage.—Mule haulage is often advantageous in low tonnage and small operations. Both horses and mules were formerly considerably used in both metal and coal mining. More or less use is still made of this method in small coal mines and in nonmetallic mineral mining where quantities handled per day are about 200 tons or less. In Table 27, W. F. Boericke gives a good summary of mule-haulage data.

TABLE 27.—MULE HAULAGE UNDER VARYING CONDITIONS

	No. 1	No. 2	No. 3	No. 4
Average tons per shift	180.0	288.0	400.0	256.0
Average distance trammed feet	1,450.0	2,613.0	2,000.0	1,483.0
Number of mules	4.0	9.0	6.0	2.0
Ore tons per mule	45.0	32.0	66.0	128.0
Miles traveled per mule	12.0	15.8	8.3	18.1
Ton miles per mule	12.0	15.8	25.0	36.3
Cost per ton, cents	7.9	10.9	5.4	2.8
Cost per ton-mile, cents	30.0	22.4	14.2	10.1

Belt Conveyors.—Sectional conveyors of trough, spool, and flat-belt types are of special design for underground service in coal mines. The head end of the conveyor, which contains the drive, is a unit, as is also the foot end. A telescopic section is placed next to the foot end to provide for belt tension, and two intermediate sections of graduated height to secure vertical discharge height adjoin the discharge end. Side construction is such as to prevent spillage. A detachable belt connector enables belt sections to be inserted to correspond to sections added to increase the length of the conveyor. Gathering conveyors are 26-, 30-, and 36-in. belt widths and of such lengths as to meet requirements. Room or entry driving conveyors are used in lengths up to 500 ft., belt widths 18-, 20-, and 24-in., with loop take-ups to provide belt storage for extension. Face conveyors are self-contained units, 16-in. belt and 11 ft. over-all length, equipped with ¾-hp. motor drive, capacity about 25 tons per hr., total weight 660 lb. They are often used in tandem.

Belt-conveyor speeds range from 100 to 400 ft. per min. Capacity involves belt width, speed, and the type of material to be transported. They may be operated on adverse slopes up to 18 to 22 deg. Shallow mines can sometimes be served by main conveyors that deliver from the principal axis of the development layout to the tipple at the surface.

Relatively little use had been made of conveyors in deep metal mining. Probably the principal use is to take the discharge from an underground crusher station and to deliver it to skip-storage bins at a shaft station or to transfers serving a haulage level. Belt conveyors are sometimes installed in shallow mines and open pits that are served by underground workings, to deliver ore from loading points and transfers to main conveyors that in turn deliver to surface bins. At the Spruce open pit (Minnesota) 4,500 ft. of belt conveyors, 3,000 ft. underground, elevate ore 387 ft. from the low point in the drifts beneath the pit to the shipping bins on the surface. The conveyor system operates at 500 ft. per min. and delivers 750 tons per hr. At the Sullivan mine, Kimberley, B. C., a 36-in. belt conveyor, in six sections totaling 2,538 ft., was installed between the 3,350- and 3,900-ft. levels to convey ore from the lower level to the main haulage adit.

Long-distance belt conveyors were installed by the H. C. Frick Coke Co. in 1924. A belt system 4¼ miles in length and with a capacity of 1,250 tons per hr. was installed for delivering coal from underground workings. It consisted of 19 units from 321 to 2,439 ft. in length with belt widths from 48 to 60 in. and speeds from 350 to 500 ft. per min. A recent surface installation at the Shasta Dam (California) consists of 25 units, aggregating 9.6 miles in length, capacity 1,100 tons per hr.; belt speed is 550 ft. per min. The system delivered from gravel pits some 12,000,000 tons in 4 years. Cost is said to have been 17 cents per ton.

Where relatively large tonnages must be delivered day by day over comparatively short distances and elevations are not too great, belt conveyors frequently offer advantages over locomotive and train transportation as well as over autotruck haulage.

Chain Conveyors.—Chain conveyors consist of shallow sectionalized metal troughs 12, 15, and 20 in. in width in which a chain, with flights attached at intervals, is drawn, passing over a sprocket at the discharge end and returning in a pan beneath the trough. A compact head drive and a tension tail-sprocket section with intervening sections make up the conveyor. Chain speeds are 90 to 135 ft. per min. Capacity varies with the speed and with the type of material; a 15-in. chain conveyor (15-hp. motor) delivers 48 tons per hr. for a distance of 300 ft.; a 75-ton-per-hr. conveyor requires a 20-hp. motor for a length of 300 ft.

Chain conveyors are used principally for coal and nonabrasive minerals. Short elevating conveyors of the chain type are used to receive the discharge from shaking conveyors for delivery to cars. The pit-car loader makes use of the chain-conveyor principle, and conveyors on some loaders are of the chain type. Chain conveyors can be shifted, taken apart, and moved, although not so conveniently as the shaking-pan conveyor.

Shaker or Shaking-pan Conveyors.—The shaker-pan conveyor is a sectionalized metal trough supported at intervals by rollers; it is given a reciprocating motion by a motor-operated drive, one stroke being rapid and the corresponding return being slow, the result being to move the material in the trough in one direction. The stroke can be adjusted up to a length of 11 in., and the degree of acceleration may also be adjusted; the number of strokes ranges from 60 to 82 per min. and is adjustable to accord with the nature of the lump material to be moved. A 5-hp. motor is required for a No. 1 trough 200 ft. in length, a 7½-hp. motor for a length of 300 ft. The maximum conveyor length is from 300 to 500 ft.; greater distances are handled by using conveyors in tandem. The conveyor discharge end can be elevated so as to load cars or to deliver to a belt conveyor. Chains attached to eyebolts in the roof of a working are sometimes used to support the trough; swivels and angle troughs enable changes in direction to be made, but straight runs are preferred.

Conveyors are loaded by hand or by means of a duckbill attached at the head end. The duckbill is a scoop-shaped telescopic end attachment to the trough, taking its motion. It is swiveled at its attachment to the trough and can be swung upon an arc, being advanced and retracted by the operator. An automatic duckbill is available for thin seams, but for thick seams the feed is manually controlled. A smooth bottom seems to be necessary for its proper functioning. Thus the shaking conveyor with its duckbill attachment becomes a loading and transportation unit that can be lengthened by the introduction of additional sections as the face of a development opening is advanced. It is widely used in coal mining, especially in thin seams.

The shaker conveyor is used to some extent in metal mines, in iron and copper mining, for short horizontal transfers from face to ore chute, or from finger raise to vertical transfer. It is also used in salt, gypsum, and potash mining. Its accessory, the duckbill, has not found application in metal mining, as floor surfaces are too rough and irregular. Highly abrasive ores are not suitable for the shaker conveyor; its field is the softer ores and some industrial minerals. It is not suitable for very wet and sticky ores. It can be used for moving material upgrade, but it is more generally used on downgrade movement.

Shuttle Cars.—The shuttle car is a rubber-tired vehicle with low top and bottom clearances. The bottom of the car is provided with a flight-conveyor rig, independently driven, which advances the loaded material when loading and discharges it when unloading. In effect it is a short horizontal conveyor mounted upon an underslung truck. It is powered by storage battery, electric cable and reel, or by double trolley. Travel speeds are 3½ m.p.h. loaded and 5 m.p.h. empty; wheel gage is 68.75 in., over-all width 79.25 in., capacity 5 tons for a height of 42 in. and 6 tons for a height of 48 in. A smaller unit has a height of 32 in. and a capacity of 2½ tons; another, 38 in. high, has a capacity of 3½ tons. Capacities given are for coal. The unloaded weight of the 42-in. car with battery is 6.4 tons; that of the 32-in., 5.4 tons.

Shuttle cars were originally designed for use with mechanical loaders to span the gap between face and main haulage or between face and main conveyor run in coal mining. By their use, room tracks can be dispensed with; their adoption originated the term "trackless mine." Shuttle cars have also been used in mining zinc ore, potash, salt, gypsum, and limestone. Other designs than the end-discharge type have appeared, notably a bottom-discharge type. Shuttle-car applications are increasing.

L. E. Young gives an example of a tread-tractor loader serviced by two 7-ton shuttle cars in a potash mine. The time to load a car was 50 to 60 sec., to unload 60 sec.; total time to load 85 mine cars (467.5 tons) was 205 min.; average distance hauled in shuttle cars was 200 ft.; three working places were cleaned up, the shuttle cars loading into mine cars.

Rope Haulage.—Endless-rope haulage comprises a two-track system with an endless rope operated by sheaves and motor drive at speeds ranging from 2 to 4 m.p.h. Loaded cars are hauled on one track and empties are returned upon the other, the cars being spaced at regular intervals. Attachment and detachment of the cars is effected automatically by rope grips. Endless rope haulage has been used for surface and underground transportation in both coal and metal mines and is still used to some extent, though seldom in American mines.

Tail-rope haulage is single-track haulage in which a train of cars is hauled by a wire rope and a double drum hoist, the tail rope being attached to the end of the train and returning about a sheave at the end of the run; the train of empties is hauled by the tail rope on the back trip, the main haulage cable being attached to the rear of the train of empties and being pulled out to the far terminal. Rollers placed at intervals support the cables and reduce wear and friction. The tail rope, double the length of the main cable, is carried from its drum by sheaves at the side of the haulageway to the terminal sheave. Curves and undulating grades can be overcome. Speeds are from 6 to 10 m.p.h.

The engine plane is an incline up which loaded cars are pulled by rope and hoist, the empty car being attached and lowered from the top of the plane. The gravity plane is a two-track incline of 20 to 30 per cent grade, served by a two-drum hoist. The descending loaded car lifts the empty on the incline, the hoist controlling the speed of lowering. In other designs of gravity planes, a single or double sheave equipped with heavy band brakes is placed at the head of the gravity plane and is used to control operation.

Rope haulage is used in both coal- and metal-mining operations but has been superseded by locomotive haulage to a large extent. The engine and gravity planes are necessary where grades exceed the limits for locomotive operation, but in modern mining they are avoided whereever possible.

Locomotive Haulage.—The electric locomotive of the trolley and storage-battery types dominates underground horizontal transportation in mines of 500 tons or more output per day. Many small mines nevertheless make use of the storage-battery locomotive. Gaseous coal mines impose severe limitations upon the use of the electric locomotive, state laws prescribing conditions under which it may or may not be used. Other kinds of locomotives, such as the gasoline and diesel types, have been used underground, but only where unusually favorable conditions prevail and ventilation is sufficient to maintain a safe dilution of exhaust gases, which contain more or less carbon monoxide. Compressed-air locomotives have some advantages over other types in gaseous coal mines; steam-storage locomotives have also some application in short runs to surface terminals. Both diesel and gasoline locomotives are sometimes used in adit haulage where ventilation is good and state laws are not strict; they are also sometimes used in large-section tunnel construction. The rack-rail electric locomotive, once used where severe adverse grades prevailed, is now seldom seen.

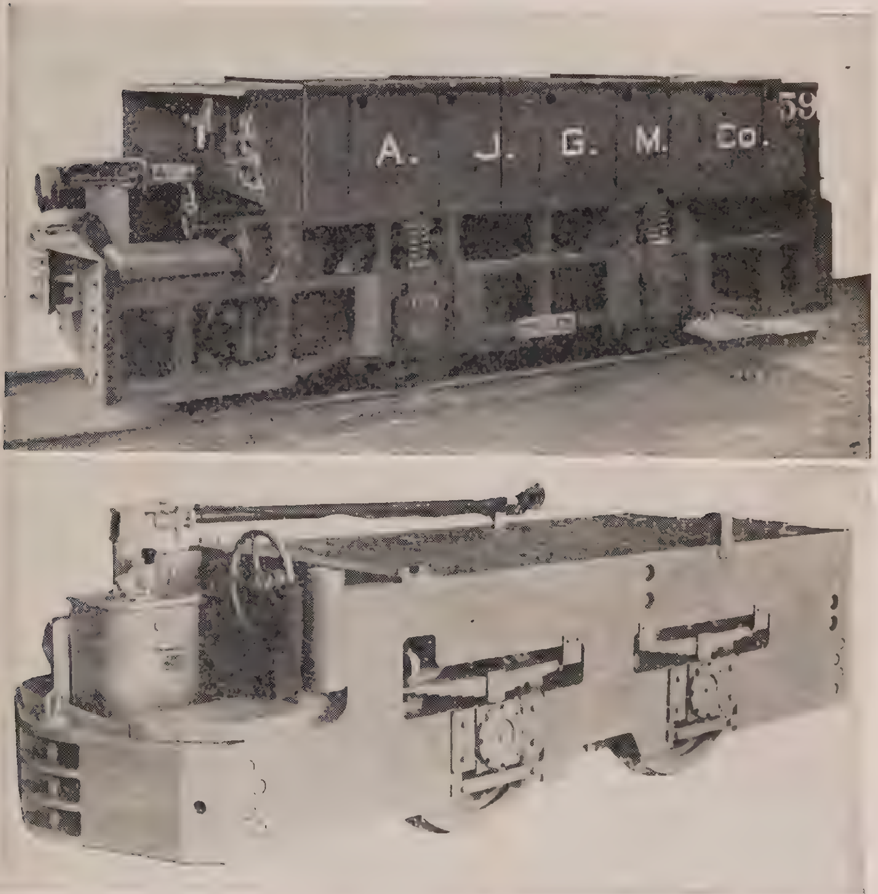


Fig. 83*a*.—Electric mine locomotives of trolley and storage-battery types. (*Westinghouse Electric & Mfg. Co.*)

The electric locomotive is of compact design and low in height. The outside-frame type prevails. Motors are d-c and usually of 250 volts.

Locomotives range from 2½ up to 20 tons in weight. The number of motors depends upon the size of the locomotive, ranging from a single 20-hp. motor in a 2½-ton locomotive to two or three 120-hp. motors in a 20-ton. Sizes range from 3 to 13 tons in metal mining; where larger locomotives are necessary two smaller locomotives are operated in tandem. In coal mines locomotives up to 20 and 40 tons are used in main haulages. Coal-mine gathering locomotives are of the cable-reel, storage-battery, combined trolley and storage-battery, crab (wire-rope haul), and combined crab and cable-reel types, ranging from 3 to 8 tons in weight.

Storage-battery locomotives are of the same general design as the trolley type. The storage battery is in a flat steel box, usually equipped with rollers to facilitate removal to charging stands and placed upon the frame of the locomotive. Batteries are of the Edison and lead-plate types with over-all capacities up to 150 kw.-hr. Battery locomotives used in haulage are comparable in weight to trolley locomotives for similar service. Smaller battery locomotives are used in gathering service; permissible types are used in gaseous mines. The battery locomotive is favored in metal mines for transportation upon other than main haulage levels, where the trolley type is generally used. A small storage-battery "trammer" finds widespread use in exploration and development workings in metal mining. Its specifications are as follows: Weight 3,000 lb., minimum length 47.5 in., maximum length 71 in., wheel base 24 in., height 41.5 to 52.75 in., speed 3 m.p.h. loaded and 6 m.p.h. light, battery 8.1 to 13.5 kw.-hr., and track gage 18 to 24 in. Storage-battery locomotives do not require bonded rails, but charging stations consisting of motor generators, charging panels, and charging stands are necessary accessories. In place of motor generators, ignitron rectifiers are coming into greater use for battery-charging stations and for supplying direct current for trolley-wire circuits.

Drawbar pull is 25 per cent of locomotive weight for steel-tired wheels and 20 per cent for chilled cast-iron wheels; on dry sanded rails drawbar pull increases to 40 per cent of weight. Capacity of locomotives is estimated by taking a factor of 30 lb. per ton for level track and an additional 20 lb. per ton for each 1 per cent of grade. The speed of haulage locomotives is 8 m.p.h., of gathering locomotives 6 m.p.h.; on long hauls and heavy service, locomotives are designed for 10- to 12-m.p.h. speeds.

Electric locomotives are equipped with bumpers having pockets for drawbar connections corresponding to the height of the car hitchings and bumpers, guard lugs being also provided to prevent car bumpers from climbing. Headlights and horn signals are part of the equipment. Controls are of the drum type; large locomotives have full or semimagnetic control; where two locomotives are used in tandem, a single control for both units is provided. Trolleys are offset on one side of the locomotive; sometimes pantograph instead of pole types are used. Hand brakes are provided. Westinghouse supplies a hydraulic brake equipment with separate motor drive, oil pump, and accumulator.

Layout of Haulage System.—Wherever possible, main shafts, adits, and main entries (coal mines) are so placed as to symmetrically divide the area to be exploited. This general plan reduces transportation distances. Level development in a deep mine also stems out from either side of crosscuts that connect to the centrally placed shaft. Traffic zones are to either the left or the right of the main traffic stem. A flat coal seam is divided at intervals by cross entries to the right and left of main entries, and panel entries stem from the cross entries at intervals and are parallel with the main entry. A pitch seam is opened by a slope of two or more main entries upon the dip; cross entries are at right angles and follow the strike of the bed. Deposits of salt, potash, phosphate, and borax and massive ore deposits also admit of development symmetrically planned to minimize haulage distances. It is not always possible to develop a mineral area on this basis, as ore bodies may be erratically distributed and their dimensions and position may not be accurately known in advance of planning.

The cross section of development workings, car and locomotive sizes, the number of cars to the train, power supply and trolley lines, tracks, grades, switches, turnouts, passings, train control, and signal arrangements are details requiring consideration. Chute and mechanical loading involve chute spacing and clearances of various kinds. Main haulage levels are planned for longer periods of operation than working levels. The position and design of vertical transfers connecting working and main haulage levels require special consideration. Most important of all are the total daily tonnage to be handled and its position of origin relative to adit, entry, slope, or shaft, both horizontally and vertically.

Levels and crosscuts are constructed wherever possible with grades favoring loads, 0.75 per cent for hand tramming and 0.5 per cent for mechanical haulage. Adits are also planned with grades favoring loads. Where adverse grades are necessary they are kept as low as possible. Grades as high as 5 per cent can be serviced by electric locomotive, but main haulage grades should not exceed a 2 per cent adverse grade. Good engineering of grade layout in mine development reduces locomotive sizes and increases haulage efficiency.

Track gages range from 18 to 30 in. in metal mines. Butte, United Verde, Homestake, and many other mining districts employ the 18-in. gage; in others the 24-in. gage is standard; industrial minerals and coal mining employ wider gages, 30 in. and upwards. In southeast Missouri, the St. Joseph company uses 24-in. gage throughout its mines with but one exception, where a 30-in. gage is employed; average length of haul is about 2 miles and daily tonnage 22,000. Salt and potash mines use gages of from 36 to 42 in.

In prospecting and small-scale mining, where hand tramming prevails, rails as light as 12 lb. per yd. are common, but 16-lb. rails are generally used for hand tramming and mule haulage. Light locomotives require 25-lb. rails; heavier locomotives, 30-, 35-, 40-, 45-, and 50-lb. rails. For large tonnages and heavy locomotives 60- to 70-lb. rails are employed. Locomotive weight rather than car size determines rail weight. The St. Joseph Lead Co. uses 60- or 70-lb. rails with No. 5 welded frogs for main lines (15-ton trolley locomotive) and 30-lb. rails with No. 3½ frogs for headings (N. A. Stockett).

Wooden ties are in general use. On some main haulages creosoted and treated ties with tie plates are employed. Steel ties have many advantages but are not used extensively. Main-line haulage tracks are carefully graded, aligned, and ballasted where high efficiency and long service are required. Table 28 gives details of a number of metal-mine haulage installations.

Table 28.—Metal-mine Haulage

Mine	Bunker Hill & Sullivan	Phoenix	Consolidated (British Columbia)	Park-Utah	Inspiration	Mascot
Car weight empty, lb.	2,900	11,970	5,500	3,450	4,000	3,000
Car weight loaded, lb.	10,100	23,970	18,300	11,450	14,000	7,000
Cars in train	18	4, 5, 8	30	17	20–25	8–12
Train-load weight, tons	90	48, 60, 96	274.5	100	140–175	28–42
Track gage, in.	24	30	36	18	30	24
Minimum curve, ft.	25	25	75	21	60	50
Maximum grade, per cent	0.33	1	0.8	1	1	1.5
Average grade, per cent	0.25	0.5	0.5	0.25	0.4	0.5
Locomotive weight, tons	4	7.5, 10	6, 11, 12	2, 4	8, 10	6
Distance, ft.	10,000	900	......	10,000	4,300	2,240
Distance, maximum, ft.	......	5,800	8,000			
Round-trip time, min	60	12–20	25	......	40	20
Size opening, ft.	8 by 8	8 by 8, 8 by 9	7 by 9, 9 by 9	6½ by 5	8 by 6	7 by 7, 8 by 8
Daily tons	1,400	3,000	5,500	1,000		
Journals, type	Outside	Inside	Inside	Inside	Outside	Outside
Height above rail, in.	49	69	54.5	50.5	58	39.5
Coupling	Pin-link	M.C.B.	M.C.B.	M.C.B.	M.C.B.	Pin-link

Minimum radius of curves, measured to outside rail, is controlled by the wheel base of locomotive or car, usually the former. Car length and width, coupling space, and width of haulageway as well as clearances are taken into consideration in selecting the minimum curve radius. In hand-tramming drifts, slick sheets and turntables may be installed, but curves of 10- to 12-ft. radius are preferred to these devices, at one time quite common.

Trolley-locomotive haulage requires rail bonding and cross bonding at intervals. Electric welding of rails on main haulages is also practiced where underground temperatures are more or less constant. Trolley wires are installed at a minimum height of 7 to 7.5 ft. above the rail, either above the track center or at one side of the haulageway. As all trolley wires are a hazard, they are boxed at all dangerous points. Wheel trolleys have been displaced by glider contacts, a graphite lubricant being applied to the trolley wire at intervals by a special trolley pole mounted upon the locomotive.

Most main haulages are single track, so that passing tracks and a signal system are necessary where two or more trains are operated. Signal systems range from block circuits with red and green lights, controlled by manually operated switches, to an automatic block system. Partings or passing tracks are placed on main haulage lines at main terminals and where distances are over several thousand feet at one or more points. The number of trains operating, their speed, tonnage to be handled, and loading and dumping time are all taken into consideration in arranging train schedules. At shaft stations track layouts are either a simple tail track or a run-around, the loaded cars being dumped into shaft bins without uncoupling; a double track with crossover where cars are loaded upon cages; or double tracks on both sides of the shaft with loaded cars approaching from one side and empties being discharged on the other. Here grades, kickbacks, and car hauls are provided for returning the cars to a siding, where the train of empties is made up for return to the workings.

In coal mines, partings are located close to centers of production and are usually at more or less standard distances, 800 to 1,200 ft. for mule haulage and 800 to 2,000 ft. for locomotive haulage. In metal mines, parting tracks are unimportant except at shaft stations. In unused development workings or drift extensions, storage for empty cars is provided by spur tracks.

Track and switch accessories are standardized in a given mine. A complete switch consists of switch points, frog, connecting rods, slide plates, ground lever, and guardrails. Switches are either the simple throw switch of the ground type or a switch stand with points and frog standardized. Frog number varies with the angle of the frog, equaling the over-all length of the frog divided by the sum of the spreads at the heel and toe of the frog.

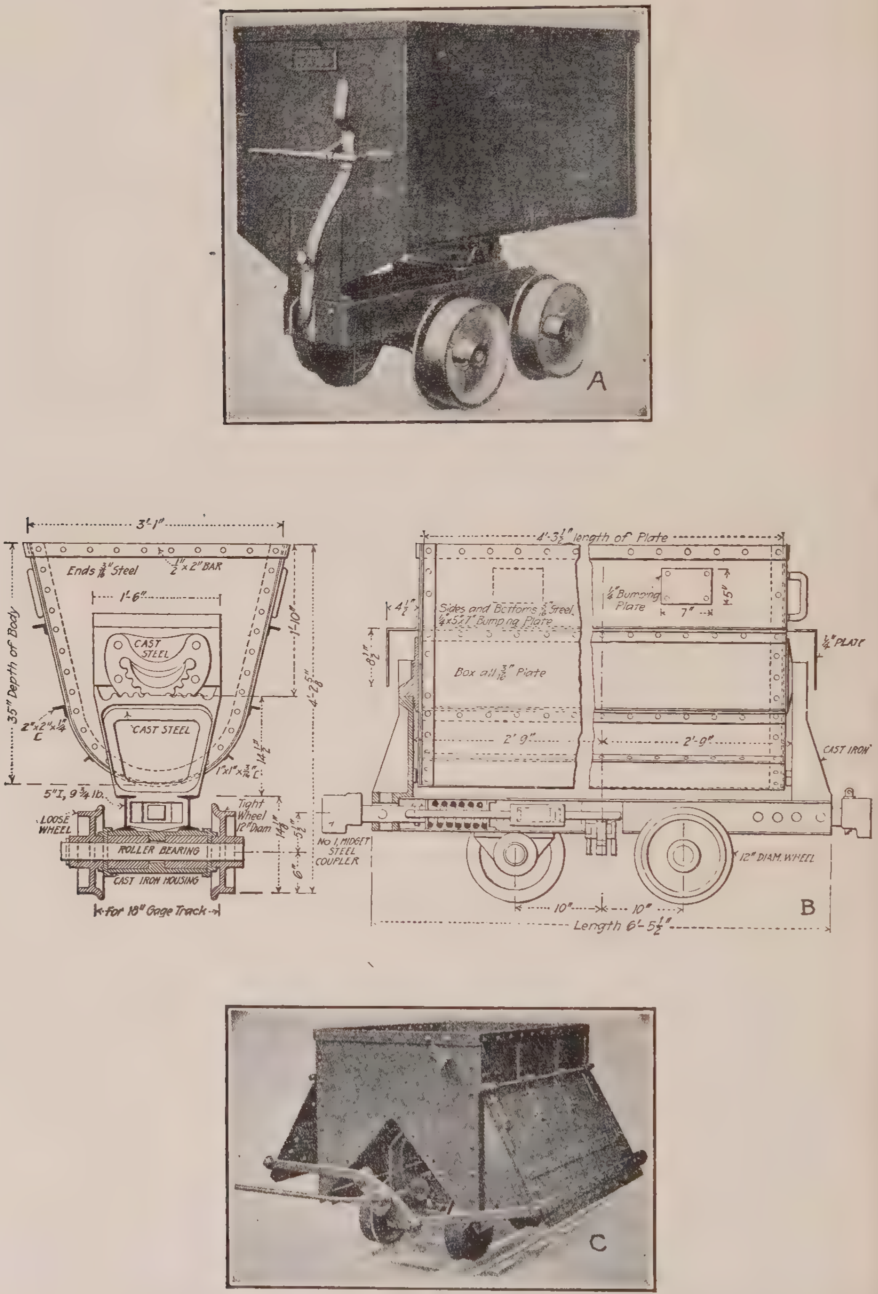


Fig. 84.—Types of mine cars. (A) End and side-dumping car; (B) rocker side-dumping car; (C) gable-end side-dumping car.

Cars.—The restricted cross section of development workings limits ore cars to relatively small sizes. The rough handling that such cars receive necessitates good design, good materials, and relatively heavy construction. Metal-mine practice requires at least three car sizes: A small car, 14 to 16 cu. ft.; an intermediate size, 20 to 25 cu. ft.; and a large size, 40 to 200 cu. ft. The small car is used in exploration, preliminary development, and small-scale mining, the intermediate size in development and mining on secondary levels, and the large car on haulage levels and adits. Many car designs are in use, some of them specialized for the kind of mining or for unusual mining conditions. Common types (Figs. 84 and 85) are the end dump, or end and side dump, the doorless or box type, the rocker or side dump, the gable-bottom side dump, the Granby side dump, and the bottom dump. Special-purpose cars, such as the drill-steel car, timber, explosives and man cars, are part of the mine equipment and are designed to meet the special conditions of individual mines. Cage dimensions determine dimensions of all mine cars where shafts are used.

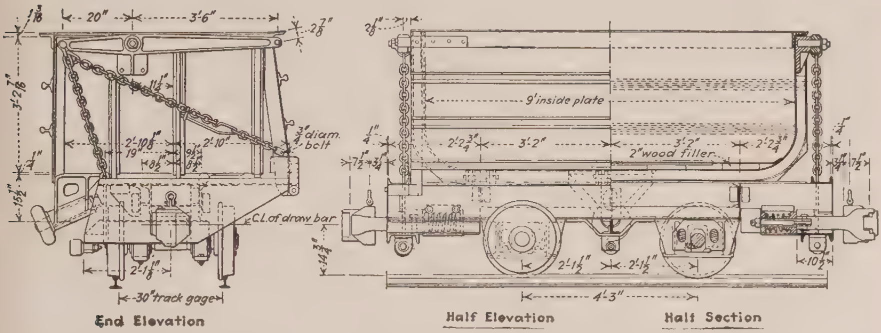


Fig. 85.—Granby-type mine car. (*A. H. Hubbell, Eng. Min. Jour.*)

Car trucks have four wheels, but some of the larger cars are supported on two four-wheel trucks. Wheel diameters range from 12 to 16 in., and wheel gages are from ½ to ¾ in. less than track gages. On most small cars wheels turn upon the axles, but on the larger cars they are integral with the axles. Roller bearings with grease reservoirs—Hyatt, Timken, and others—are generally used. Plain bearings give a resistance of approximately 30 lb. per ton of weight; roller bearings, from 12 to 15 lb. Bearing design and provision for lubrication are important factors. Bearings on large cars are either between or outside the wheels. Wheel bases range from a minimum of 16 in. for small cars up to 4 ft. or more for the large cars; a long wheel base gives better stability but requires a greater radius of track curvature; a short wheel base enables cars to be more readily maneuvered on slick sheets and in hand tramming.

Small cars are sometimes handled in trains, link-and-pin or hook couplings being used; large cars are provided with drawheads, bumpers, and link-and-pin couplings; main haulage cars are provided with spring drawheads, M.C.B. couplings, automatic couplings, or link and pin. The link and pin is now being generally displaced by the M.C. B. and automatic types.

Most mine cars are springless, but larger cars are now designed with outside journal boxes and springs. Brakes are provided on cars used where track grades are greater than normal. In a few large installations cars are equipped with air brakes. Car bodies are of steel except in some coal mines where both steel and wood are used. Steel cars are often designed with a composite bottom, a steel wearing plate on a 2-in. plank bottom where the impact of large pieces of heavy ore must be compensated. Cars and car fittings are both standardized for metal, coal, and mineral mines.

The cross section of mine workings, track curvatures, allowable clearances, and chute-lip heights determine the height and width of cars, the desired capacity being obtained by increasing the length of the car body. Small mine cars of 16 cu. ft. have box dimensions of 2 by 2 by 4 ft. and above-rail height of 42 in.; end-dump cars of 25 cu. ft. capacity are 32 by 28 by 48 in. and 46 in. above rail. They are used in drifts of 5 by 7 ft. Haulage cars vary greatly in dimensions. At Park-Utah a 68-cu. ft. car weighing 3,450 lb. has a width of 3 ft., an over-all length of 12 ft. between coupling centers, and a height of 50.5 in. It is used in an adit 5 ft. in width by 6.5 ft. high, on 18-in. gage, 40-lb. rails, minimum radius 21 ft. At the Phoenix mine (British Columbia), a Granby-type car, 130 cu. ft. capacity, has an over-all width of about 6 ft., a height of 69 in., and a length of 12 ft. 7½ in. between coupling centers. It is used in an 8- by 8-ft. and 8- by 9-ft. haulageway of 30-in. track gage, minimum radius 25 to 40 ft. The Bunker Hill and Sullivan (Idaho) bottom-dump haulage car, 40 cu. ft. capacity, has an outside width of 44.5 in., an over-all length of about 7 ft., and a height of 49 in. It is used in an 8- by 8-ft. haulageway of 24-in. track gage with minimum radius of 25 ft.

Dumping accessories are seldom required except with Granby and box-type cars. Granby cars are tipped automatically by a roller engaging a track on the opposite side of the dump, which affords a smooth curve from the point of roller contact to the highest level of lift and return to contact level. Usually some kind of limit guard is required to prevent cars from tipping too far. At the Carlton tunnel (Colorado) a 3-cu. yd. car of the Granby type was designed with a retractable roller. Doorless cars require a motor or compressed-air-cylinder-operated revolving tipple for dumping. The tipple is a cylindrical cage that rotates on rollers and is of sufficient length to accommodate one or more cars. It is designed to meet mine and car conditions and is standardized. Usually cars are uncoupled for dumping, but some tipple designs provide for the dumping of coupled cars (swivel couplings) and for the passage of the haulage locomotive. Dumping horns are used at dumps for handling end-dump cars. Large side-dump cars require compressed-air cylinders, either suspended above or installed at one side and below the car body, to assist in tilting this car body.

The doorless car has so many advantages that at one time it seemed to be the solution to most haulage problems, but present trends are away from the tipple dump and to door types of cars. The Granby car has steadily grown in favor, as it enables trains to be handled at fair speed without uncoupling. Sticky ores present a problem, and provision must sometimes be made for barring and hammering car bodies to dislodge corner and bottom accretions. Some cars are designed with rounded corners and spreading sides, which favor the discharge of sticky ores.

Weights per cubic foot of capacity depend upon car design and thickness of steel plates. They range from 31 to 68 lb. for end-dump cars; 40 to 100 lb. for doorless cars; 50 to 85 lb. for rocker cars; 45 to 80 lb. for gable-end cars; 65 to 120 lb. for Granby cars, and 50 to 73 lb. for bottom-dump cars.

TABLE 29.—PERFORMANCE AND COSTS OF LOADING BY SCRAPER LOADERS AT EAGLE PICHER
(*S. S. Clarke*)

	Sheet ground	Sheet ground L	M bed
Labor	$0.156	$0.097	$0.217
Repairs and maintenance	0.065	0.060	0.092
Power	0.014	0.026	0.015
Casualty insurance	0.001	0.001	0.002
Total	$0.236	$0.184	$0.326

NOTE.—Loading into cans, 158 tons daily average; into 1½-ton cars, 116 tons; into trucks, 142 tons. L and M refer to different mines; air-shovel operating costs ranged from $0.224 to $0.309 per ton.

Cost of Underground Transportation.—Hand-shovel loading and hand tramming for an average distance of 609 ft. gave an average cost of 22.8 cents per ton in Michigan copper mines; chute loading and hand tramming for distances 100 to 1,000 ft. gave costs ranging from 4 to 20 cents per ton. Animal transportation in various mines gave the following average: Length of haul 1,625 ft., ton-miles per mule-shift 19.8, cost per ton 7.2 cents, and 27 cents per ton-mile. Trolley-locomotive haulage for distances up to 1 mile cost approximately 5 to 8 cents per ton-mile; up to 2 miles 5.3 to 5.7 cents per ton-mile; and for more than 3 miles, 1 to 3.4 cents per ton-mile. The foregoing figures are condensed from data gathered by M. J. Elsing. Some costs for scraper-loader operations are given by S. S. Clarke and are presented in Table 29. Haulage costs in coal mining vary over such a wide range that no adequate presentation can be made here.

Transportation costs include loading, tramming, dumping, and idle time. Chutes should be designed for fast loading. Tramming time is a function of distance and speed. Speed is important where distances and tonnages are large. Dumping facilities should be such as to secure minimum delays. Switching facilities should be adequate. Less equipment or a greater tonnage with a given equipment, as well as lower unit costs, may be obtained by more efficient use of time and labor.

Surface Transportation.—Delivery to stock pile, mill, or treatment plant, to shipping point from adit, shaft, entry, or slope requires some form of surface transportation. In small-tonnage operations a variety of surface methods may be employed, but in large-tonnage operations railroad transportation is brought as close to the mine outlets as possible. The plan for the exploitation of a large mineral property includes provision for a low-operating-cost transportation link between mine and railroad transportation. Where treatment plants are more advantageously located at distant points, the connecting transportation link is carefully planned for minimum operating cost. Wherever possible such plants are located close to mine or pit outlets so as to keep transportation costs at a minimum. Adverse grades are avoided, and grades against the return of empties are kept to a minimum. Distance, elevation, tonnage per day, type of ore or mineral, and intervening topography are all considered.

The simplest case that arises is the delivery of ore to a stock pile. Assuming a shaft, skips dump into hoppers from which cars are loaded. A track system connects with trestles, temporary or permanent, over the stock-pile area. Small tonnages per day are handled by hand tramming; large tonnages, 500 tons or more, are handled by gravity tramming with rope pullback and end-dump or side-dump cars; grades are 2.5 to 2.75 per cent in favor of loads; speeds are 450 to 500 ft. per min. and pullback speeds 500 to 550 ft. per min.

Larry cars are also used to deliver ore from skip bins to stock piles or to railroad cars. Another simple arrangement is from skip to receiving bin to rock crusher, which discharges into another bin from which the ore is transported by car, conveyor, train, truck, or wire-rope tramway to the treatment plant. Up to 100 tons per day is handled by a single 2-ton car hauled by a mule or small gasoline locomotive; distances seldom exceed 500 to 1,000 ft., and mill position is such that an 0.5 per cent grade is in favor of load. For tonnages of 200 to 500 per day, a belt conveyor connects the crusher station to the mill bin. Where greater distances intervene, small tonnages are transported by autotruck, and large tonnages, 5,000 or more, by locomotive and train where topography is favorable for track location. Where track construction involves excessive cost, road construction and truck haulage can often be substituted.

Where topography makes road construction costly, the wire-rope tramway often becomes the connecting link. Difference in elevation and intervening topography determine road, track, and tramway installation costs. Roads cost from $3,000 to $15,000 per mile; track $5,000 to $20,000, and tramways $30,000 to $65,000 per mile. To overcome 1,000 ft. of elevation would require about 10 miles of track at a 2 per cent grade, about 4 miles of road on a 5 per cent grade, and 1 mile of tramway on a 20 per cent grade. The wire-rope tramway can cope with more difficult topography than the road or track system. Maximum track grades are limited to about 4 per cent, road grades from 6 to 8 per cent, and tramways to about 30 per cent.

Wire-rope tramways consist of track cables supported by towers over which buckets are drawn by a moving cable at speeds of 350 to 500 ft. per min., one cable supporting the loaded buckets and the other the returning empties. Buckets are loaded from chutes by hand and automatically dumped at the receiving terminal. Tramway lengths range from 1,000 ft. up to 5 miles or more; capacities are from 100 to 2,000 tons per day; operating costs are 20.6 cents per ton-mile for a 1-mile length and 100 tons per day; 9.5 cents per ton-mile for 100 tons per day and 5-mile length; 3.7 cents per ton-mile for 2,000 tons per day and 1-mile length, and 1.5 cents per ton-mile for 2,000 tons per day and 5-mile length (F. C. Carstarphen). Differences in elevation determine whether power will be required for the operation of the tramway or whether a steep gradient in favor of loads requires special methods for speed control.

The jig back is a short two-bucket tramway used where elevation with respect to distance is relatively great and is sufficient for the loaded bucket to pull up the empty one. A grip pulley is used for the control of speed in lowering. Its capacity is small and can be increased only by increasing the size of the buckets. Though rarely employed, it is sometimes very useful where the head of a road is near but far below the mine opening.

The combination of autotruck and road is an elastic system, in that capacity can be increased by increasing the number of trucks. Roads where autotrucks are to be used for the movement of large tonnages over a considerable period are carefully engineered with grades, tangents, widths, curves, and surfaces well within the limits of the equipment to be operated. Low maintenance and operating costs are obtained by smooth road surfaces well kept up.

Truck transportation is growing in favor in mining districts and competes with all other forms of transportation except where tonnages exceed 5,000 tons per day or more and distances are great. It is best applied to short hauls and small to moderate tonnages. In open-pit operation, road and truck transportation are displacing locomotive and track systems more and more, in both stripping and mineral mining.

Truck sizes for small tonnages range from 1½ to 3½ tons with capacities about doubled by means of heavier springs and oversize tires. Truck sizes are carefully selected for large capacities and continuous operation. Maximum sizes range from 35 to 50 tons load. Diesel-engine drives are generally used. Contract trucking is common in many mining districts. E. D. Gardner in *Information Circular* 6,898, *U.S. Bur. Mines*, gives the following ton-mile costs: Up to 1 mile, 35 cents; 1 to 2 miles, 22 cents; 2 to 5, 12 cents; 5 to 10, 9 cents; and 10 to 20, 6 cents.

Miscellaneous Transportation Methods.—Some Western mining districts were served by team and wagon. Pack animals continue to be used in places inaccessible to roads and serve under prospecting and exploration operations. Remote areas in Canada developed water transportation for summer conditions and tread-tractor sled trains for winter transportation. In late 1916, 3,300 tons of ore was hauled 40 miles by sleigh, 130 miles by barge in summer, and then by rail to Trail, B. C., a distance of 1,200 miles. In 1918, 6,000 tons was shipped, and in 1919, 15,000 tons. In 1919 as many as 300 teams were used on the 40-mile sled haul, average load per team 6½ tons, and cost 37.5 cents per ton-mile. In 1917 a tread tractor was tried out. Since that time the tread-tractor sled train has found a distinct field in winter transportation. At Flin Flon some 25,000 tons of freight was moved 69 miles in 90 days (1928-29); sleigh loads were 15 to 38 tons, average train load 75, maximum 120 tons; average time for the 69-mile trip was 23 hr., return trip 14.25 hr. (4 to 7 m.p.h.), with 8.25 hr. at both terminals; road cost was $7,000 per mile, maintenance $300 per mile, and cost of freighting 22 cents per ton-mile, exclusive of depreciation. Cost of equipment for hauling and maintenance was $202,000 (G. E. Cole).

Alaska Juneau made use of 600-ton self-dumping barges for waste-rock disposal. Freighting by airplane began in Canada about 1925, and since then many mining districts have made use of airplanes for transporting prospecting equipment and personnel for initial exploration and operation. In 1931-32 the Bulolo Gold Dredging Co. transported by airplane 2,500 tons, including parts for two complete 10-cu. ft. gold dredges and a power plant, from Lae, New Guinea, to Wau over an intervening mountain range, a distance of 32.5 miles, at a cost of $762,000.

HOISTING

Methods of Ore and Coal Handling.—Cage hoisting is used for handling men, materials, and loaded ore or coal cars. Three types are prevalent. In the first the cars are placed upon and taken from the deck of the cage by hand; in the second, caging machines are used at loading points and decaging machines at the surface; in the third, a tilting-deck cage enables cars to be dumped without removal from the cage.

Cage hoisting is used in some degree in all shaft mines. Supplies, materials, and men are best handled by means of cages. Only in small mines is the cage used for hoisting ore, but in many coal mines cage hoisting is used to avoid excessive coal breakage and dust. An end-dumping mine car is automatically locked upon the deck of the tilting cage; as the cage approaches the dumping point it is automatically tilted, the end door of the car is lifted, and the contents is discharged into a chute. At such mines a caging machine is used at loading points. Caging and decaging machines are rarely used in metal or industrial mineral mines but are widely used in coal mines.

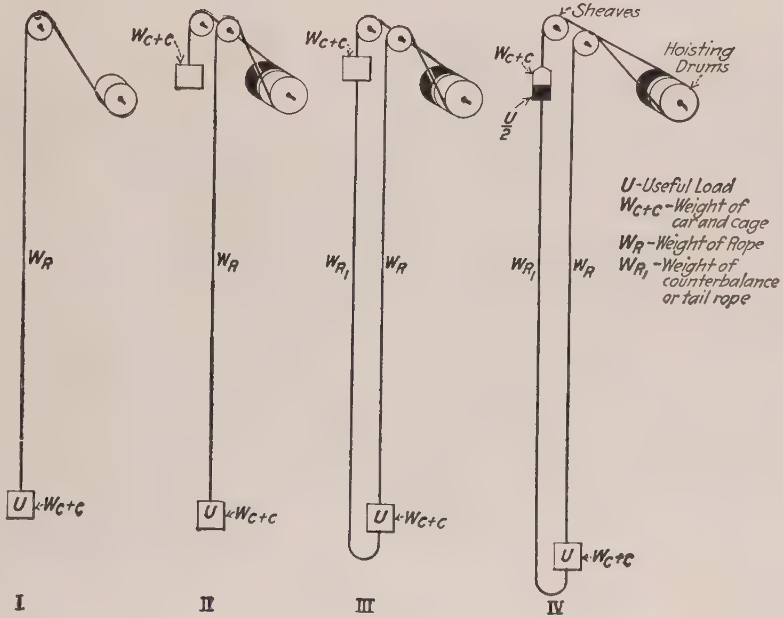


FIG. 86.—Systems of hoisting.

Skip hoisting is almost universal in all large metal mines and for most industrial mineral mines served by shafts. It is used to an increasing extent in coal mines. Skips are loaded at shaft bins, usually from supplemental measuring pockets that hold a single load, and are automatically dumped at the surface into bins. Shaft capacity is greatly increased by skip hoisting. Both skip and cage hoisting are required in mines of large output, and separate compartments or separate shafts are provided for these services. Shaft service may be required at a number of levels in deep mines but may be necessary only at one level in coal mines.

Systems of Hoisting.—Four systems of hoisting are illustrated in Fig. 86: Unbalanced or single-rope hoisting, partially balanced or two-rope hoisting, balanced or two-rope hoisting with tail rope, and overbalanced hoisting. Single-rope hoisting is characteristic of small operations and prospects. Slope mines, coal and iron, also make use of this system where inclines are 30 deg. or less. Most auxiliary hoisting is on a single-rope basis.

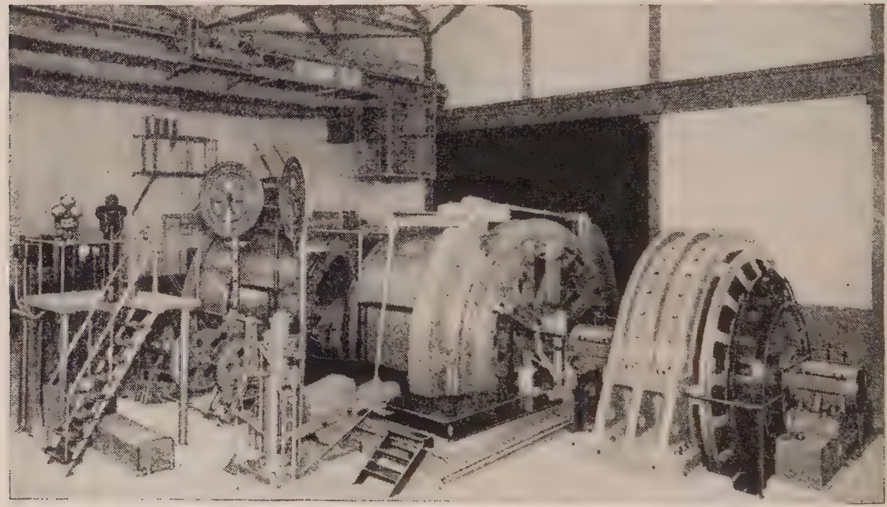


FIG. 86*a*.—Two-drum, motor-driven hoist; rope pull 56,800 lb; maximum rope speed 2,250 ft. per min.; 2,000 hp. d-c motor, direct-connected, 59.5 r.p.m.; manual and automatic control. (*Allis-Chalmers Mfg. Co.*)

The two-rope partially balanced system is more used than any other in both metal and coal mines. The two-rope type with tail rope is comparatively rare. In Germany the Koepe hoist is used in deep coal mining. It consists of a 15- to 25-ft. sheave, deeply grooved and directly connected to motors; the rope is looped about the sheave and its ends are attached to the cages in the compartments; a tail rope is connected with the cages beneath. Grip sheaves of somewhat smaller diameter also were used. B. V. E. Nordberg described the Bollen pulley, a type of grip sheave, and its use at the Magma mine (Arizona). At this mine, where the total lift is 4,000 ft., a tail rope is used to achieve balance. A skip and cage are in balance with a three-deck cage, the useful load being 8,000 lb. At the Park-Utah a similar hoist is installed directly over the shaft. In each case both sides of the system must be used simultaneously.

An overbalanced system was at one time used in small mines and prospects to avoid overloading small power plants. It is now common in handling large cages, but the tail rope has been discarded. The counterbalance and its rope require a separate small compartment in the shaft. Their weight is equal to the weight of the cage, and the overbalance weight is usually half the useful load. Starting with a loaded cage at the top, the excess load in favor of the hoist is half the useful load.

Vertical and Incline Hoisting.—Hoisting in incline shafts differs from vertical hoisting in that there is increased friction and a decreased proportion of the weight to be hoisted. Rollers are placed at intervals of some 50 ft. to support the rope and to reduce wear. Tracks are necessary. Hoisting speeds are one-half to less than two-thirds those employed in vertical shafts of corresponding depths. In slope hoisting, trains of mine cars are handled instead of skips or cages. In double-track slopes a train of loaded cars is hoisted while a train of empties is being lowered, the hoisting ropes being attached and detached as required; in single-slope hoisting a train of loaded cars is hoisted and the rope is detached and fastened to a train of empties, which is then lowered. In metal-mine slopes skips are used in place of trains.

Types of Hoists.—Electrically operated hoists have displaced steam- and compressed-air-driven hoists for both large and small outputs. Hoists operated by internal-combustion engines are used in prospecting operations where electrical power is not available. Water power is seldom used.

Hoists fall into two general types, direct-driven and gear-driven. In the former the motor is direct-connected to the shaft of the drum; in the latter, single or double reduction gear trains are interposed between motor and drum shaft. Hoists are either single or double drum. Drums are cylindrical, conical, cylindroconical, and bicylindroconical. The conical drum provides a rope wind beginning at a minimum diameter and increasing in diameter with each successive turn; the cylindroconical begins the wind on a cylindrical part that connects with the spiral path of the conical portion; the bicylindrical begins the wind on a cylindrical portion, continues it on a conical portion, and completes the wind on a cylindrical part. The rope follows a continuous groove on the various surfaces. Single drums are keyed to the drum shaft; double drums have one drum keyed and the other friction-clutched or toothed-clutched to the shaft, or both drums are friction- or toothed-clutched. In the tandem two-drum hoist the drums are mounted upon parallel shafts and connected to them by friction clutches, the parallel shafts being connected by a gear train to the motor. The tandem hoist gives smaller fleet angles than the two-drum hoist of the usual construction.

Because the hoisting rope is wound upon the drum under maximum load, the drum design must be such as to resist the crushing action of the coils of rope. Sometimes from two to five layers of rope are wound upon the drum, and under such circumstances the drum must be of massive construction. Cylindroconical and large drum hoists are designed for a single layer of hoisting rope. A two-rope single-drum hoist may consist of a central cylindrical section into which two conical end sections lead, one on each side. The drum is keyed to the shaft, and both ropes use the cylindrical section as either completes its wind.

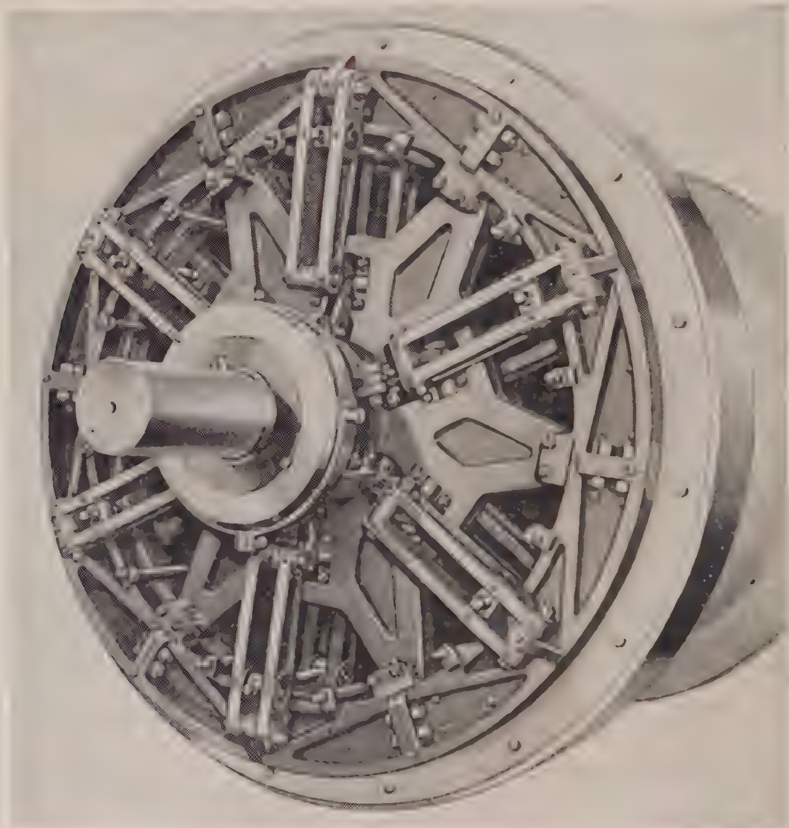


Fig. 86b.—Multiple-arm double-disk clutch for hoist drum. (*Allis-Chalmers Mfg. Co.*)

Friction clutches (Fig. 86*b*) are widely used and are especially designed for the heavy service of hoisting. The jaw clutch is positive but does not permit sufficiently close adjustment between the positions of the two drums and corresponding cages. A multiple-toothed clutch is used on some large hoists and permits faster and closer adjustment. It is favored in English hoisting practice, while the friction clutch is favored in this country.

Very small hoists are equipped with band brakes, but all large hoists have post brakes on each drum, designed for parallel movement and bearing upon brake wheels of sufficient diameter and face width to control movement of the drums. Application of post brakes is accomplished by compressed-air cylinders or by weights lifted by oil or compressed-air cylinders. Supplementary methods of control include dynamic braking in large hoists. Such control or countertorque reverses the function of the motor, which then becomes a generator.

Both d-c and a-c motors are used for hoisting service, the latter being of the slip-ring type. The wound-rotor induction motor with secondary resistance control is used for many small hoists and up to 2,000 hp.; but as it has high starting peaks and is not completely controllable, it is not well adapted for lowering. D-c motors are usually employed on large hoists.

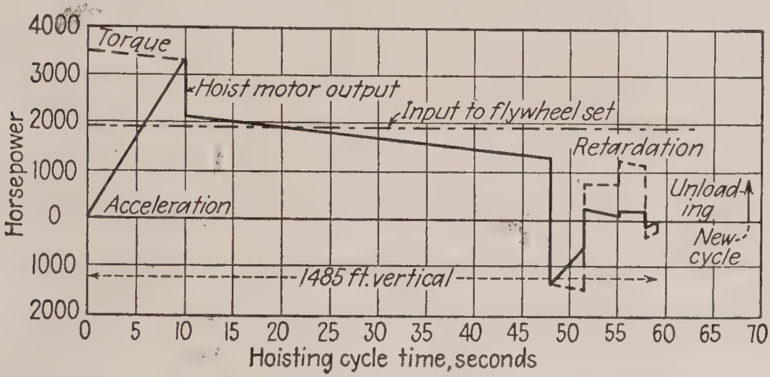


FIG. 87.—Power demands in theoretical duty cycle curve for balanced hoisting from 1,485-ft. level. (*F. R. Grant, Eng. Min. Jour.*)

In the operation of a hoist, power demand (Fig. 87) rapidly ascends to a peak during the accelerating period of 5 to 20 sec., and then declines, rapidly at first and more slowly during the constant-speed interval until retardation begins, at which point input power becomes zero and mechanical or dynamic braking, or both, absorb the momentum of the moving parts and bring the cages or skips to rest at predetermined points. The most adverse conditions occur in unbalanced hoisting where load, skip, and rope have to be accelerated; in two-compartment hoisting, the unbalanced weight is only that of the load and rope, but at the retardation period the momentum of the descending rope must be absorbed, as well as that of the ascending load and the pair of revolving drums and accessories.

Control of speed and prevention of overwind are effected by manual or by manual and automatically operated devices. Automatic devices are mechanical or mechanical and electrical.

Overload Power Control.—In hoist operation, peak power demand occurs during acceleration. Power demand alternates from peak to no load for each cycle of hoisting. Motor-generator flywheel units with Ilgner-Ward-Leonard control are used to supply current to the d-c hoist motors. A wound-rotor induction motor with slip-ring regulation drives a d-c generator and heavy flywheel continuously. When the hoist motors approach their peak demand, energy stored by the flywheel is taken by the system, momentarily reducing flywheel speed; during retardation motors act as generators and return energy to the system. It is possible by this arrangement to maintain power input from the a-c supply at approximately constant load. Ward-Leonard control, or variable voltage control, comprises series connection of hoist motor and generator, the control of the motor being obtained by reversal and variation of the generator-field excitation. Field currents of the hoist motor and generator are controlled by magnetic contactors. Automatic regulation of hoist operation and control has been achieved by this combination. At Inspiration, Miami, and relatively few other mines, hoisting is initiated by switches at skip-loading stations underground and automatically terminated at skip dumping points.

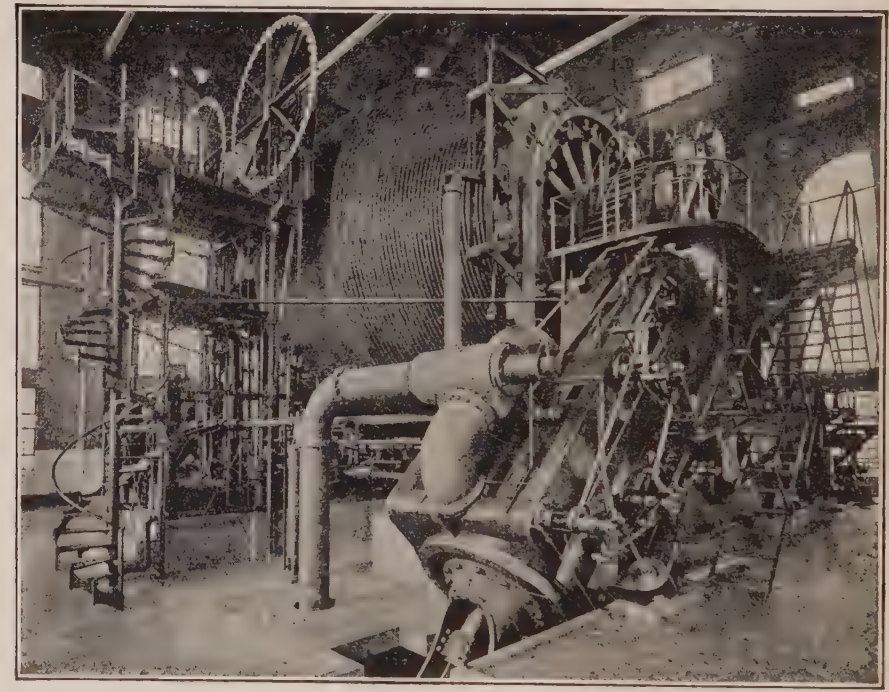


FIG. 88.—Types of hoisting engines—the Quincy hoist.

A modified Ward-Leonard system in which the speed and the rate of acceleration are automatically controlled by a group of small pilot generators is known as the Regulex system. G. A. Morrison states that the auxiliary motor generator consists of six machines coupled to a common bedplate, which, with a master controller, a switchboard, a main motor, and a main motor generator, makes a complete unit. Hoisting from a depth of 2,000 ft. is being accomplished without a hoistman. The Amplidyne is still another device (General Electric) for hoist-motor regulation. It regulates speed, voltage, current, or power over a wide range where d-c motors are used.

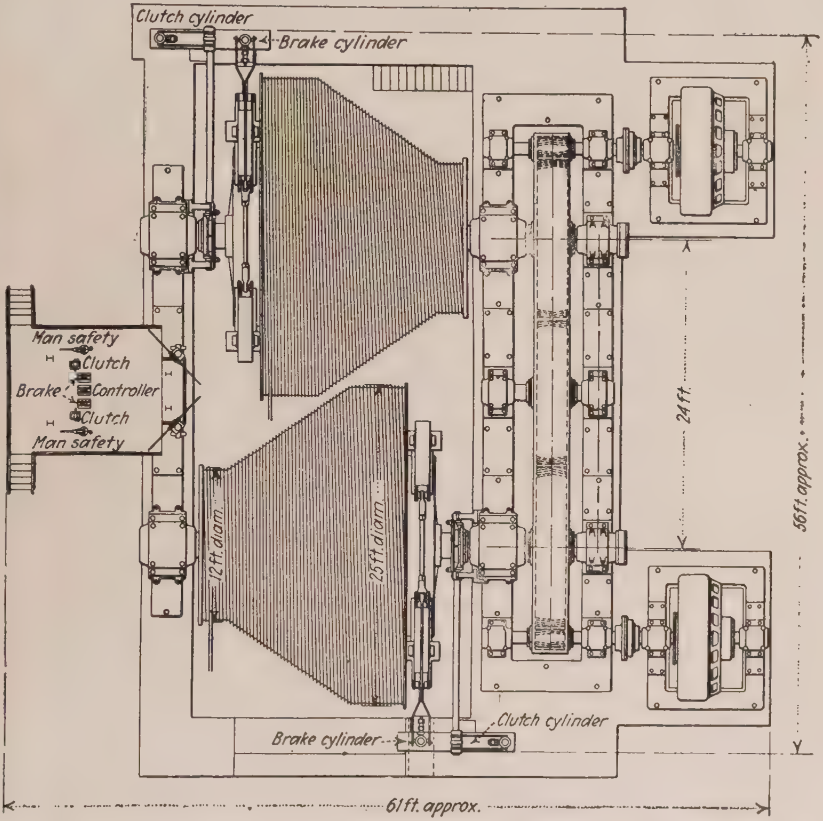


FIG. 89.—Plan of Homestake bicylindroconical tandem drum hoist. (*Eng. Min. Jour.*)

Examples of Hoists.—Partial data are given in the accompanying table for five large hoists operating on large properties. The equipment is representative of present practice for deep mines. In the deepest operation, the Simmer and Jack mine (South Africa), a steam turbine is used to drive the flywheel generator supplying direct current to the hoist motors. The capacity of the hoist is rated at 21.6 trips per hour from a depth of 6,660 ft.; acceleration is 17.6 sec., followed by a second acceleration period of 17.6 sec. to obtain the maximum rope speed of 3,000 ft. per min.; total trip time is 2.685 min. with 5-sec. intervals for loading and dumping. In contrast to this, the Ross shaft hoist shortly after it was installed handled some 150 trips, a total of 1,050 tons, in 4 hr., or at the rate of 37½ trips per hour. Figure 88 shows a direct-connected steam hoist and Fig. 89 the plan of a gear-driven electric hoist.

TABLE 30.—PARTIAL DATA FOR LARGE HOISTS

	Porphyry shaft, Inspiration mine	Porphyry shaft, Inspiration mine	Ross Shaft, Homestake	Vlakfontein, South Africa	Simmer & Jack, South Africa
	Ore hoist	Man and material hoist			
Number and type of drums	2 C	1 C	T 2 Bi-C-Con.	Bi-C-Con	2 Bi-C-Con.
Diameter of drum, ft.	12	10	12–25	13–35	13–35
Face of drum, ft.	8 G	8.5 G	17 G	11 G	
Rope diameter, in.	1⅞	1½	1⅞	2	2
Rope length, ft.	1,700	....	5,400	6,200	
Rope weight, per ft.	5.35	....	5.63	....	6.89
Rope speed, ft. per min.	1,920	800–1,400	2,500	3,000	3,000
Skip capacity, tons	12	....	7	8	8
Skip weight, tons	10.5	....	6.25	6.81	4.5
Motor, hp	D-C 2,150	D-C 600	(2) D-C 1,500	(2) 1,810	
Motor connection	Direct	Geared	Geared	Geared	Geared
Maximum drum speed	51 r.p.m.	....	35 r.p.m.	....	65.3 r.p.m.
Generator, d-c, kw	1,600	450	Two 1,250	Two 1,430	
Motor, a-c, hp	1,900	500	1,750	3,300	
Flywheel, tons	39	8.25	43.75		
Service, tons	10,000	....	4,000		
Service interval, hr.	16	....	15		
Depth of wind, ft.	1,485	....	2,900	5,240	6,600
Acceleration, sec.	10	....	15	....	2–17.6
Clutch	Friction	....	Axial plate	....	Toothed
Control	W.-L.	W.-L.	W.-L.	W.-L.	W.-L.

NOTES.—C is cylindrical; T is tandem arrangement of drums; Bi-C-Con. is bicylindrical conical drum; G is grooved; D-C is direct current; A-C is alternating current. All hoists listed are equipped with post brakes operated by gravity, with power cylinders for release. Safety equipment of American hoists is Lilly controls; W.-L. is Ward-Leonard control.

The foregoing comprises hoists for serving vertical shafts. A single-drum, single-rope hoist, serving a slope at Ensley, Ala., has a drum diameter of 18 ft. by 17-ft. face, the rope diameter 1½ in. and length 7,450 ft.; weight of load is 15 tons, skip weight 10.4 tons; motor horsepower is 2,500; speed 3,600 ft. per min.; slope length 6,600 ft. Another example is the Quincy No. 2 (Michigan), served by a hoist taking 10,000 ft. of rope in one layer, the incline being 9,000 ft. in length. The small diameter of the hoist is 16 ft. and the cylindrical portion 30 ft., both ends being conical drum.

Shaft Elevators.—Elevator hoists for service to depths of 1,000 ft. have been found adaptable for handling men and light materials. The hoist is semiautomatic, electrically driven and controlled from the cage as in the case of the ordinary elevator. Speeds are 700 ft. per min. A number of installations have been made, but 1,000 ft. is the maximum vertical depth.

Supplemental Hoisting.—Small compressed-air-operated hoists, single drum, are used in winzes and raises and for miscellaneous work underground. The stretcher-bar hoist is powered by 2- to 6-hp. compressed-air engine; drum diameter is 5 to 9½ in., face 5 to 17 in.; rope capacity 300 to 1,000 ft. of ¼- to 5⁄16-in. wire rope; capacity 500 to 2,000 lb.; hoisting speeds 60 to 100 ft. per min.; and weight 200 to 800 lb. This type of auxiliary hoisting unit was the predecessor of the larger scraper hoists described elsewhere.

Hoisting Speeds.—Hoisting speeds for handling men are regulated by law. A common speed limit is 800 ft. per min. Other speeds are determined by shaft, hoist, and headframe conditions. In the deepest shafts speeds are a maximum of 3,000 ft. per min. or more; in shallow shafts, 500 ft. in depth, the maximum is 1,200; for depths of 500 to 1,000 ft., 1,600 ft. per min.; for 1,000 to 2,000 ft. per min., and for over 3,000 ft., 3,000 ft. per min. The desired shaft capacity influences the design of the hoisting installation in respect to loading, accelerating, hoisting, retardation, and dumping time, the sum of which gives the total time for a cycle. As the hoisting time is usually greatest, the speed of hoisting and the shaft depth over which the maximum speed can be maintained determine this time interval. It is for this reason that great speeds are used in very deep shafts.

Hoisting Ropes.—Steel wire ropes of ordinary-lay, lang-lay, preformed, flattened-strand, Seale, nonspinning, and locked-coil construction are used in hoisting practice (see Fig. 90). Some steel-wire flat ropes are used but have been generally displaced by the round rope. Ropes of ordinary lay consist of six strands of 19 wires each, the rope lay (twist) being opposite to that of the strands, there being right- and left-lay ropes. Various combinations, such as eight strands of 19 wires each, six strands of 37 wires each, and others, are manufactured either for obtaining greater flexibility or for greater wear resistance. The most commonly used hoisting rope is the 6-19. In the lang-lay rope, the twist or lay of the wires in both strands and rope is in the same direction. As such ropes tend to untwist, they must be carefully handled.

Strands are laid together about a hemp core. Flattened-strand ropes are constructed of strands formed about an elliptical or triangular-shaped wire core. Preformed ropes are constructed of strands in which the individual wires are formed or given a set to correspond with the lay of the strand, so that they remain in place and do not untwist. Preforming is applicable to most constructions and makes for ropes of greater flexibility which can be more easily handled, which do not kink so readily as ropes of ordinary construction, and in which broken wires do not protrude. Preformed ropes cost more. Flattened-strand and lang-lay ropes give greater wear, as more metal is exposed on the surface of the rope. With the exception of the core wires in the flattened-strand ropes, all ropes that have been described are constructed of wires of the same diameter.

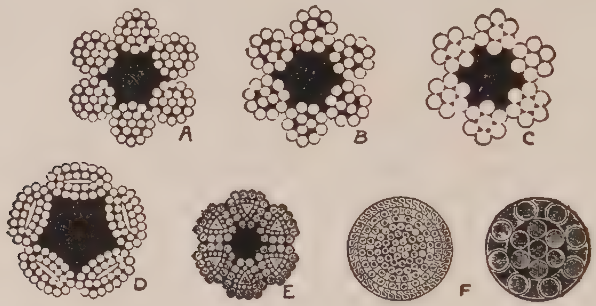


Fig. 90.—Types of hoisting ropes shown in cross section. (A) is a standard hoisting rope of six strands of 19 wires each about a hemp center; (B) is a 6 × 12 rope used for haulage purposes; (C) is a 6 × 7 guy rope; (D) is a flattened-strand rope 5 × 28; (E) is a triangular-strand rope 6 × 25; (F) locked-coil ropes, the first used for hoisting purposes and the second for cage guides.

A rope of Seale construction is made up of six strands, each of which consists of a core wire surrounded by nine inner wires of uniform but relatively small size and on the outside by nine wires of uniform size but of relatively larger diameter. The larger outer wires of each strand offer more resistance to abrasion and seat themselves in the valleys formed by the smaller wires. Some hoisting installations have used ropes of Seale construction. Warrington construction consists of six light strands, each consisting of 12 outer wires, large and small alternately, and surrounding seven wires of intermediate size.

Nonspinning ropes are usually constructed of an inner and outer rope of opposite lays. The practice of preforming wire in strands has made such ropes less used than formerly. Nonspinning ropes are used in sinking operations where the bucket is suspended below a crosshead. The locked-coil rope is a nonspinning type but is seldom used for hoisting in American mining practice.

Hoisting ropes are constructed of plow steel and improved plow steel, the approximate wire strength in pounds per square inch being respectively 210,000 to 260,000 and 230,000 to 280,000. An acid open-hearth steel is used. Strength of wire ropes and recommended loads are given in manufacturers' catalogues. Ropes of ordinary construction stretch from 1.3 to 2.3 per cent of their original length with use. Locked-coil ropes stretch the least.

Internal and external lubrication is necessary to minimize wear and to prevent internal corrosion. Wear is not only upon the outer surface but also internally upon the individual wires. Ropes fail by reason of kinking, crushing, unlaying, and bending and by dynamic loads in excess of normal working conditions, such as taking up slack too quickly or excessive acceleration. Resistance to bending, sheave-passing ability, or flexibility is in direct proportion to the number of strands and the number of wires in a strand, while resistance to abrasion and crushing is inversely proportional to the number of strands and wires in a strand. Passing about small sheaves is the most severe service to which a rope can be subjected. Preformed rope and the more flexible ropes can be used upon relatively smaller diameter sheaves than ropes of ordinary construction. It is for this reason that such ropes are used for scraper service and for the smaller hoists. For man cages and ore hoisting, with ropes of relatively large diameter, both sheave and minimum drum diameters are selected with ratios of 80 to 100 times the diameter of the rope. Minimum sheave diameters for small ropes depend upon rope type; thus a 6-7 rope would require a ratio of 48 and a 6-37 a ratio of 20 (R. A. Rairden). Manufacturers' catalogues give recommended sheave diameters for various sizes and rope types.

The condition of a rope is judged by the number of broken wires exposed upon its outer surface, by loose wires, by measurement of wear upon outer wires, by diameter shrinkage, by measurement of increase in length of lay, and by evidences of internal corrosion. It is general practice to cut off a portion of the lower end of a used hoisting rope and submit the wires to bending, torsion, and ductility tests. Sometimes the actual breaking strength of the rope is determined by submitting a short length to a tension test. Rope inspection is a daily task. More complete inspections of hoisting ropes are made at weekends. The rope must be cleaned for a thorough inspection. Preformed rope is difficult to inspect.

Life of Hoisting Ropes.—Life of hoisting ropes is given in terms of hours of service, tons hoisted per rope, or number of hoists per rope. Care taken in placing ropes upon drums, efficient lubrication, and skillful hoist operation give longer rope life and a better time range of normal safety factor. Short ropes have relatively less life than long ropes. There is less wear in vertical shafts than in inclines or slopes; the number of wraps on a hoisting drum influences wear; the presence of water and steam in a shaft is an adverse factor. Practice varies as to when a rope reaches a condition requiring replacement, but closely spaced inspections will indicate the approach to unsafe deterioration. A complete log detailing the results of inspections is kept for each rope in service.

A. F. Williams gives two examples of hoisting ropes in use from 1,552 to 5,472 hr., loads hoisted being 315,663 and 701,727 tons, depths 2,520 and 1,840, and factors of safety 7 and 5.2, respectively, with loads of 5 tons. At Hollinger a 1½-in. 6 by 19 preformed lang-lay rope hoisting from a depth of 3,050 ft. at a maximum speed of 3,000 ft. per min. lifted 1,001,811 tons of 6-ton skips in 166,968 trips before being replaced (J. W. Dougherty and J. M. Douglas). The average of five 1½-in. 6-19 lang-lay ropes in iron-mining service, each 4,020 ft. long, was 1,164,567 tons hoisted. R. B. Smart gives the life of two 2-in., 6-32 lang-lay preformed ropes, each 6,200 ft. long, as 32 and 45 months and states that a rope life of 3½ to 4 years is not unusual where large bicylindroconical hoists are in use.

Factor of Safety.—The ratio of breaking load of rope and the maximum load or combined weight of rope, cage or skip, and useful load is the factor of safety for static conditions. Bending stress, friction, and the pull due to acceleration when added to the combined weight and divided into the breaking load of the rope gives the dynamic factor of safety. Operating conditions in each case determine the dynamic factor of safety, and these are considered in the design of the hoisting installation. The rate of acceleration and the amount of slack in the hoisting rope at the beginning of a hoist cycle are important factors. Static factors of safety for hoisting men, which frequently are prescribed by law, are from about 7 to 10; for ore hoisting such factors range from slightly less than 5 for very deep hoisting to 7 to 10 for shallow hoisting.

Fleet Angle and Lead.—The angle of fleet is the angle made by the rope between the drum flange and the center line of drum and sheave. The distance between sheave and drum centers is the lead. The greater the lead, other things being equal, the smaller the fleet angle. Too great a lead results in vibration and whipping of the rope between sheave and drum. Idler sheaves are frequently installed where long leads are necessary. Fleet angles are limited to 1½ deg. for flat-faced drums and to 2 deg. for grooved drums, but some engineers place these angles at 2 and 4 deg., respectively.

Rope Attachment.—Hoisting ropes are attached to drums by passing them through a hole in the drum, looping them about the drum shaft, and securing the loose ends to the respective ropes by rope clamps. The excess length of rope is coiled around the drum and should be sufficient not only to prevent complete unwinding but also to provide for recapping during its useful life. Attachment to the cage and skip is provided by a rope socket, wire-rope clips, or various designs of wedge sockets. Wire-rope clips are probably most commonly used. In this arrangement the rope is passed around a forged thimble and pressed into the grooves of the thimble by a special clamp; the free end of the rope secured by a number of bolted clips at intervals of 6, 8, 10, and 12 in., starting at a point 4 in. from the thimble. A heavy bolt connects the thimble to the drawbar of the cage or skip. In most cases an additional fastening, consisting of from two to four chains attached to a bolted sleeve socket above the rope clips, connects with the cage. Detaching

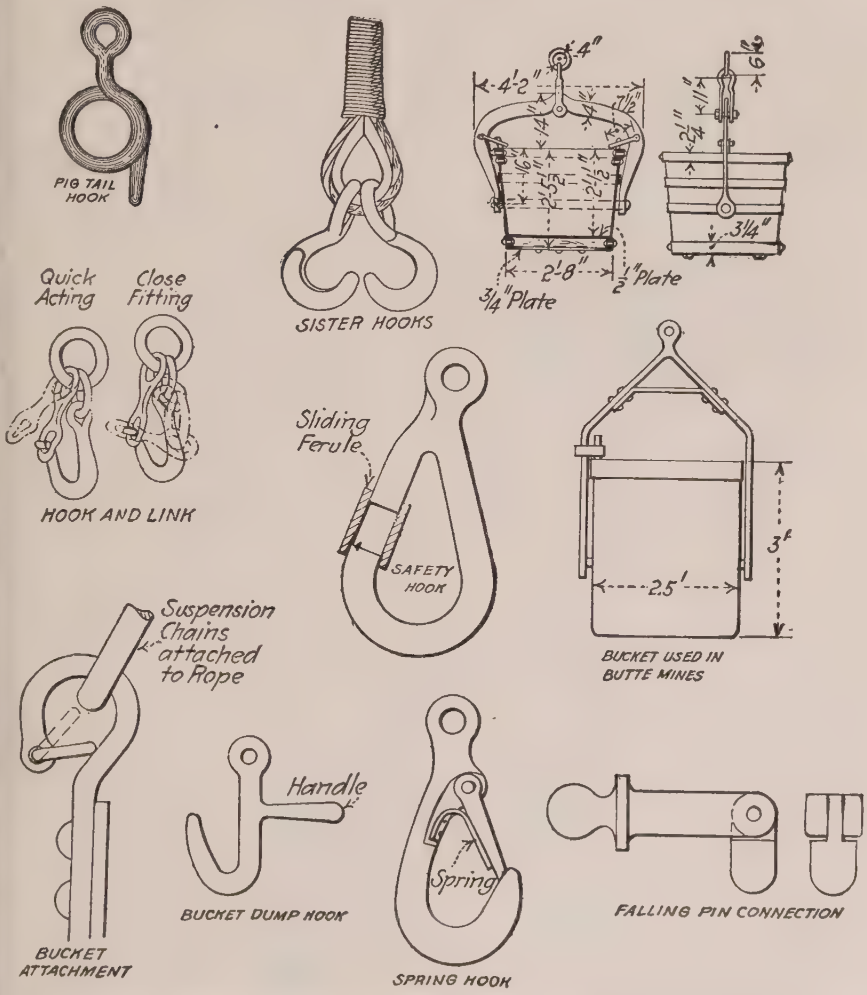


FIG. 91.—Bucket hooks and devices for attaching the hoisting rope to the bucket. Types of buckets.

hooks, interposed between thimble and cage and used to detach the rope from the cage in case of overwind, are seldom used.

Buckets, Cages, and Skips.—The vehicles used in hoisting are designed for the shaft compartments and to conform to operating conditions and the service to be rendered. The bucket, bucket and crosshead, cage, skip, or combined skip and cage are types of vehicles for vertical shafts. Man cars and skips are used for inclines, trains of mine cars for coal-mine slopes. Buckets are used in shaft sinking and in small mines. Figure 91 shows rope attachments for buckets. In the Tri-State lead and zinc districts ore hoisting by bucket is common practice. Safety crossheads (Fig. 92) provided with automatic safety catches reduce the hazard involved in bucket hoisting.

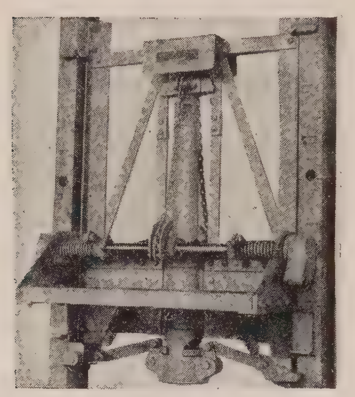


FIG. 92.—Safety crosshead for bucket hoisting. (*Joshua Hendy Iron Works.*)

Cages for the ordinary shaft compartment, 4 by 5 ft., have limited handling capacity, and special loading arrangements are required for large haulage locomotives (see Fig. 93). Long timbers are handled by throwing the cage hoods back, standing the timbers on end, and lashing them to the hoisting rope. The restricted deck capacity of such cages brought about a shaft design that provides a cage compartment 10 to 14 ft. in length with a width of 4 to 6 ft. Cages are usually single deck,

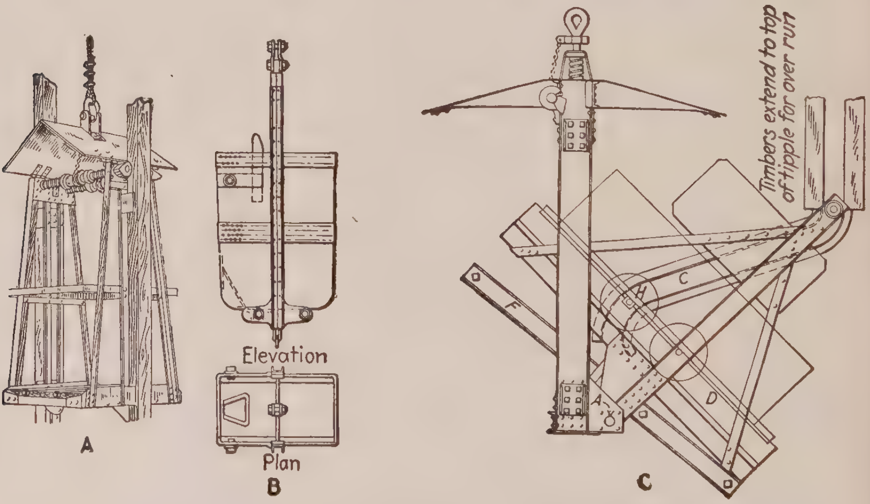


FIG. 93.—Shaft vehicles. (A) Safety cage; (B) ore skip; (C) tilting deck cage.

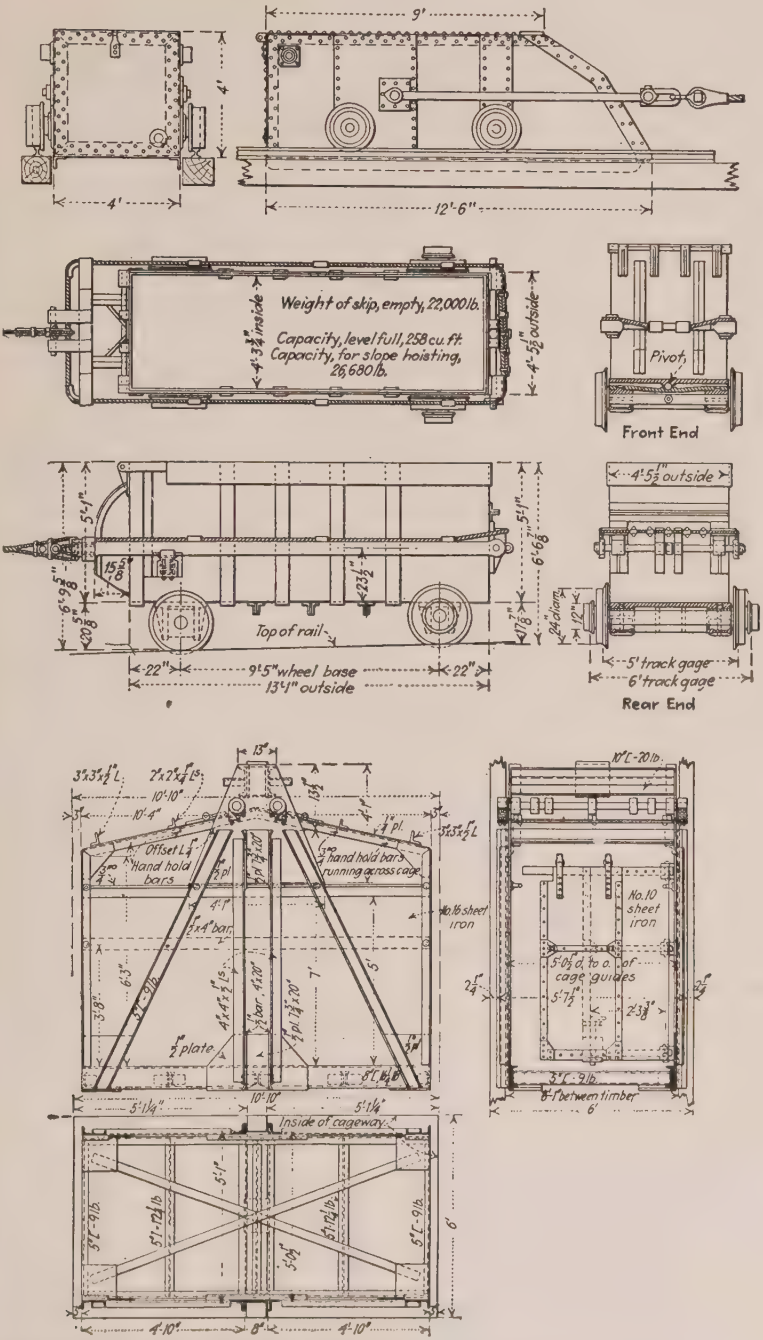


FIG. 94.—Skips for incline hoisting; below, typical large compartment cage. (*L. Eaton, Eng. Min. Jour.*)

but two-deck cages are used in mines of large output. In such large-cage compartments a single cage is operated with a counterbalance, and its rope is in a separate smaller compartment. American mines favor two compartments for handling ore in skips with a hoist for this service and a large cage and counterbalance compartments with its own hoist. In some South African mines two large cages are operated in counterbalance. Though cage sizes vary with the requirements of the mine, a deck area 14 by 5.5 ft. is large enough for most mines. Such a cage will load 50 men per deck, the allowance being approximately 1.5 sq. ft. deck area per man. Cages of this kind (Fig. 94) are frequently equipped with chairs at each corner, all operated by a lever on the cage. This arrangement overcomes the objection to the use of keps at stations.

Skips range in capacity from 1 to 17 tons, in metal mines from 1 to 10 tons, rarely 12 tons; in South Africa an 8-ton skip is generally used for deep hoisting. The cross section is square or rectangular, and the depth is 1½ to 4 times the minimum width. For serving vertical shafts the box is pivoted to a bail or frame, which is provided with a crosshead and guide shoes. Rollers on the sides of the box engage curved tracks in the headframe and tilt the box about its pivot into the dumping position. Figure 95 shows a skip dumping rig and skip. Dumping horns ride upon tumblers and enable the skip rollers to engage an upper curved track, thus completing the dump. Various devices are used to secure stability during hoisting. One of the earliest of these consisted of a latch that was lifted at the dumping point by a roller on the end of the latch engaging a track. Another design consists of an upper stop on the side of the box and lower down a half shoe, which slides along the guide opposite the upper stop plate. A space is provided through the guides at the dumping point to enable the half shoe to cross the guide as the box is tilted.

Skip design conforms to shaft and ore conditions. Abrasive ores require replaceable wearing plates or bars upon bottom and sides where the ore stream strikes in loading and discharging. Sticky ores require rounded corners; in one design short lengths of chain are fastened in each corner to loosen accretions; in another design thick rubber pads are used in the bottom to facilitate clean discharge; in still another a cast steel bottom of hemispherical shape is welded to the side plates.

For service in incline shafts the bail is attached to the box below both the mid-point and the center line. Four wheels support the box upon the rails and may be attached to the sides of the box, thus giving a low center of gravity, or may have their axles fastened to the bottom of the box (Fig. 94). Dumping is accomplished by wide-tread rear wheels, the extra tread engaging the dumping tracks and tilting the box at the discharge point. Guides are used in shafts of steep inclination but may be omitted in shafts of low inclination. In such cases stringers

(Fig. 94) run the length of the skipway, and angles are attached to the bottom of the box in such a position as to engage the stringers in the event of a derailment. Special loading hoppers are necessary for inclines of 30 to 50 deg.; for 15- to 30-deg. inclines, the top plate of the box is omitted and a gate is fastened to the bail at the front end of the box. Hoisting stability is secured by using wide track gages, up to 5 ft. Swiveling the front or rear axle in a plane normal to the tracks allows for track inequalities.

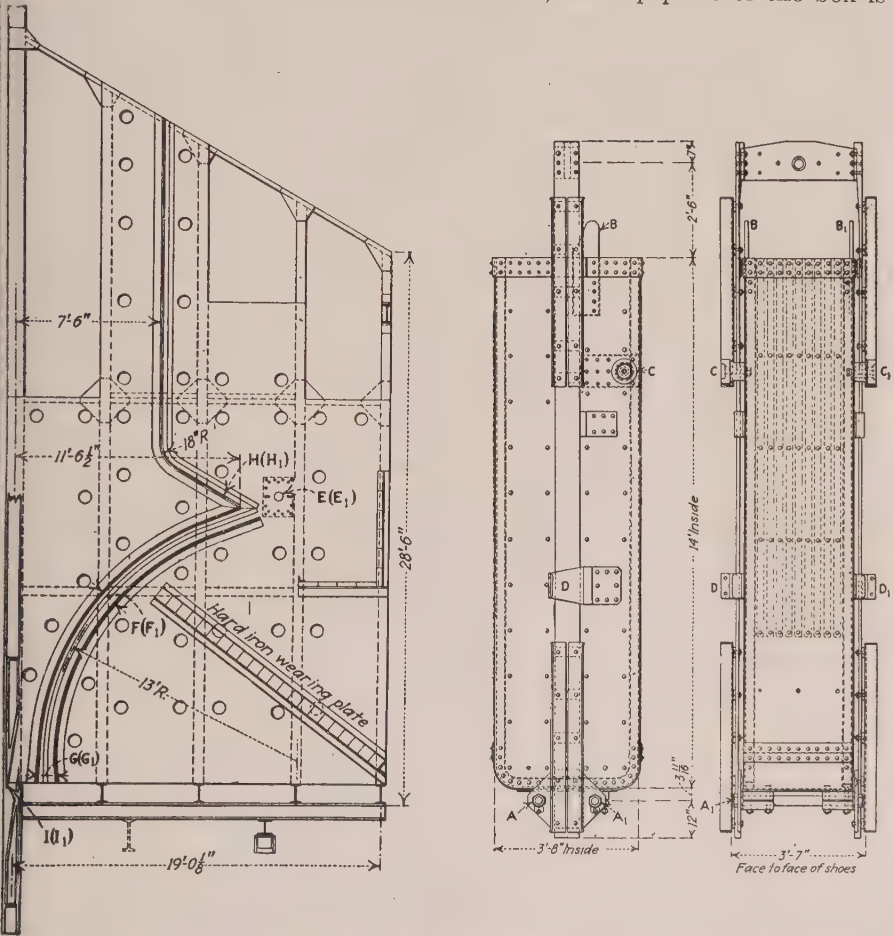


FIG. 95.—Skip dump and typical skip of the Butte district. (*C. M. Harrer and L. S. Farnham, Eng. Min. Jour.*)

Mechanically, the vertical skip has the objection that in dumping, part of the vertical weight is transferred to the dumping tracks, with the result that after dumping the skip returns slowly to its vertical position; there is also considerable shock as the skip enters the dumping tracks.

To overcome this objection, a new design (Anaconda) makes use of a side discharge at the bottom of the box and the skip is never tilted. The bottom of the skip is sloped at a 50-deg. angle to the discharge edge. A large side-discharge door operates in grooves, its width being the width of the box less that of the side strips necessary to support the grooves. In dumping, the skip is raised to a fixed position, at which point a compressed-air cylinder advances a fork or yoke that engages lugs at the top edge of the side door; the skip is lowered, the side door remaining stationary, and its contents are discharged into a side trough. There is said to be little spillage with this design; it also has other advantages over the ordinary skip dump.

Skips and cages are constructed of mild steel, design following established practice. Manganese steel is sometimes used for skip boxes and at points of concentrated abrasion. Weights of cages 12 sq. ft. in deck area range from 2,700 to 4,000 lb.; with those of deck 12 ft. 7 in. by 5 ft., or about 63 sq. ft. area, the cage weight is 8,000 lb. and load 5 tons. Skip weights approximate 1,000 to 1,500 lb. per ton of capacity for 7- to 15-ton capacity and 1,500 to 1,900 lb. per ton of capacity for 2- to 6-ton skips. Small cages and skips weigh more per square foot of deck area and per ton of capacity than do the large sizes. The 12-ton skips of Inspiration's Porphyry shaft weigh 21,000 lb., or 1,750 lb. per ton of capacity. A 5-ton (94-cu. ft.) rounded-bottom skip designed by O. D. McClure, constructed of alloy-steel plate and steel castings, weighed 6,000 lb., or 1,200 lb. per ton of capacity.

The use of high-tensile steel with proportionate reduction in the cross section of structural parts gives lower weights; the use of duralumin gives still lower weights, as much as 30 to 60 per cent weight reduction as compared to ordinary steel. At the Falconbridge mine (Canada) a 9-ton nickel-steel skip weighs 10,100 lb. or 1,010 lb. per ton of capacity; a 13,100-lb. cage accommodates 54 men or a load of 13,000 lb. At the Hollinger mine (Canada), a 30-man steel cage weighing 11,500 lb. was replaced by a 60-man two-deck cage of aluminum alloy weighing 8,265 lb. At the Cayuga salt mine a 2,800-lb. steel cage was replaced by a 1,650-lb. aluminum alloy cage. In equipping an underground shaft, the United States Smelting Co. installed an aluminum-alloy 50-cu. ft. skip weighing 1,520 lb. and a cage of steel and aluminum weighing 1,980 lb., the comparative weights in steel being 3,600 and 2,700 lb., respectively. The skip was lined with 3/4-in. rubber, over which were fastened 3/16-in. high-carbon steel plates to deflect the ore. The respective corrosion resistances of steel, alloy steel, and aluminum alloys to shaft and ore conditions require investigation in each proposed installation.

Shaft Arrangements.—Wooden guides are generally used in timbered shafts and also in most steel or concrete-supported shafts. Guides are of selected straight-grained timber dressed on all sides, 4 by 4 in., 4 by 6, 6 by 6, 4 by 8, and 5¼ by 9 in. in size. Douglas fir, pitch pine, and other similar strong woods are used. Steel channels, 8 in. by 3¾ in., and 80-lb. steel rails are sometimes installed. In circular shafts, wire-rope guide cables of 1⅝-in. diameter are used at the corners of the cages, and two 1⅝-in. rubbing cables are placed between cage runs, both guide and rubbing cables being weighted at their lower ends to ensure maximum rigidity (weights are from 5½ to 6½ tons per rope). In some circular shafts wooden guides are used in preference to rope guides. Wooden guides permit the use of safety dogs on cages and skips but must be of substantial cross section to be effective; metal guides do not permit the use of safety dogs.

Keps or landing dogs, once installed at all stations, are now practically obsolete. When they are used they are installed upon the cages as a part of the cage construction and are advanced or retracted by a system of levers as required at loading stations. Chain keps, which are short lengths of chain attached at one end to shaft timbers, the other being attached to the cage after it has reached the station, are sometimes used. They have the advantage of swinging free from the cage attachment as the cage is started upward.

Gates are installed at each shaft station to safeguard the entrance to each compartment and to prevent objects from falling into the shaft. Shaft doors are either waist high or are designed to cover the full opening of the shaft. A wooden bar is the simplest arrangement, but a wooden door or a steel-frame door with heavy mesh is a better design. Cage loading stations are equipped with tracks that extend to the shaft edge and line into the deck tracks; they are arranged to be accessible at either one or both ends of the cage compartment.

Skip loading stations are equipped with loading bins, sometimes of several tons capacity, a skip load, or storage pockets up to several hundred tons capacity. The actual skip loading point (Fig. 96) is a sufficient vertical distance under the level to provide space for the storage bins into which the ore is dumped at the level. Two general designs both arranged as side-discharging hoppers are in use. In one the discharge front is closed by a steel or timber structure in which a sliding steel gate, operated by handwheel, rack and pinion, or compressed-air cylinder, controls the flow of ore into the skip; the flow is deflected by a metal chute designed to discharge into the skip. The front structure of the pocket is recessed in the space behind the shaft timbers for a sufficient distance to provide a working platform at the control gate and to provide access by ladderway from the level above. The other design provides a large recessed space between the shaft timbers and the front of the storage pocket. This space is sufficient to accommodate loading bins for each skip, each bin receiving a measured skip load of ore from compressed-air-cylinder-controlled gates and chutes in the front of the storage pocket close to each loading bin. Loading bins are provided with quick-opening gates and chutes extending to the skip loading points. After a skip is loaded and during the interval when it is being hoisted, its skip loading bin is filled from the storage pocket and is ready to be emptied when the skip returns. The foregoing arrangement, appearing in many variations, is the most effective loading arrangement for skips in vertical shafts. With it spillage is at a minimum.

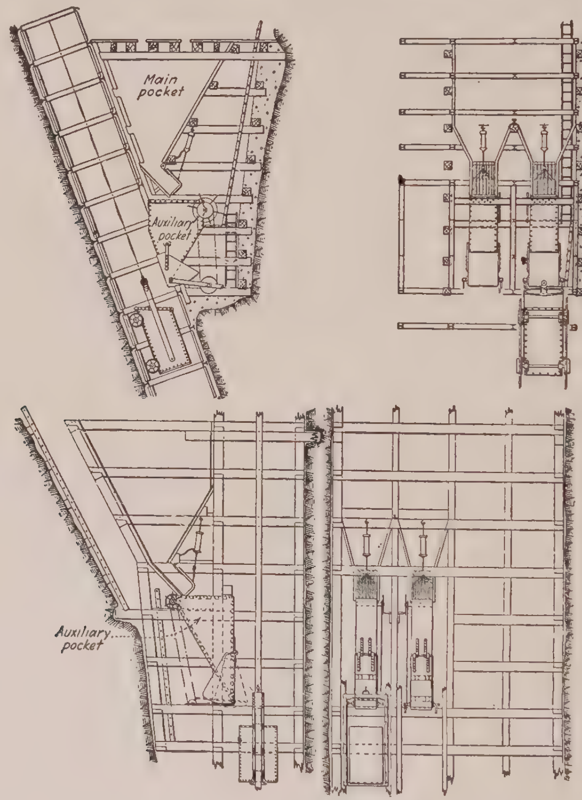
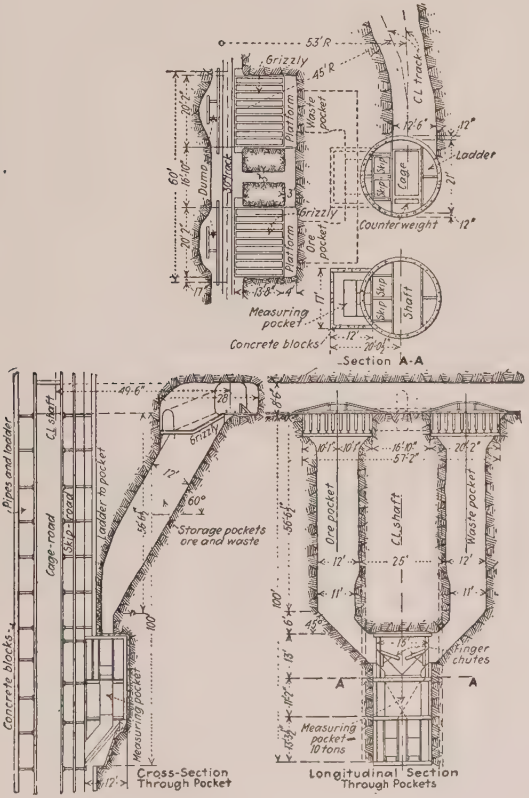


Fig. 96.—Skip loading bins for incline and

The angle of shaft inclination influences the details of design of loading stations for incline shafts to a considerable extent. The loading station is excavated in the hanging wall above the shaft; for steep inclines it follows the design for vertical shafts, but for intermediate angles special chute hoppers or funnels are required to direct the ore stream into the mouth of the skip. On low angles storage bins are omitted, and the small ore pockets discharge directly into a hopper beneath which the open-top skip is spotted.

Shaft Signal Systems.—The simplest signal mechanism is the pull rope, a $\frac{1}{4}$- or $\frac{3}{8}$-in. galvanized cable attached at the upper end to a bell crank and counterweighted by means of a spring or weight. The bell crank operates a spring-trip gong. Quarter-inch wire rope weighs about 100 lb. per 1,000 ft. The rope is placed in the corner of the shaft, close to the cage at the landing points. Buttons or rope pulls are attached to the cable for convenient operation and cleats keep the rope in position. The pull-rope system can be operated up to 2,000 ft. or more, but signaling cannot be done from fast-running cages. It is usually installed in a shaft for emergency purposes. The pneumatic system, used for one-level hoisting and for shallow shafts, consists of a closed pipe system, $1\frac{1}{2}$- to 2-in. pipe, operated by piston and cylinder, which generates sufficient pressure to operate a whistle or bell at the hoist. It cannot be operated from a moving cage.



vertical shafts. (*L. Eaton, Eng. Min. Jour.*)

In deep shafts and large-capacity hoisting installations, some form of electrical signaling is employed. One type consists of a single charged wire with return through the cage and hoisting rope. Contact is made by a metal rod connected electrically to the cage or skip, or by a pull switch on the cage. A 22-volt current is used. This rig is used in inclines and slopes. A more modern version consists of a bare conductor supported by insulators in the shaft compartment and connected at its upper end to a sensitive telegraph relay, the other contact of which is connected to a circuit with contacts on the hoisting drum and return through the rope to the cage. A 3-volt L-20 Edison storage battery is placed upon the cage and connects with a hoist signal box making contact with the shaft conductor. Signals are given from the cage. The relay operates the hoist bell system. H. D. Keiser states that such a signal system has been operating satisfactorily in the Park City district (Utah) over a period of years.

The modern shaft is equipped with an independent system of signaling and wiring for each hoisting compartment. Indicator lamps are placed upon the cage or skip position indicator to show where signals originate. In addition to the bell signal, a lamp indicates the signal to the hoist operator by flashing. Pull switches are placed at each shaft station and can be operated by the cager from his position on the cage. A call circuit with electric horn signals is also operated by pull switches at each station. This enables the cage to be called to a given station. Hoist signals are given only by the cager. Signals for skip hoisting are given by the skip loader at the loading station. Pull-rope signals are also installed for emergency use. A telephone circuit with transmitters and receivers at each station and at one or more places in the shaft house is used as an auxiliary.

The signal system just described does not provide for signaling safely from a fast-moving cage. To overcome this objection, the Homestake mining company devised and installed a carrier-current system. This makes use of the hoisting rope and radio apparatus for one-way conversation from the moving cage to the hoist loud-speaker or from the hoist to the cage loud-speaker, a switch at the microphone being operated by either the cager or the hoist engineer when communication is desired. High-frequency current, generated by an oscillator tube and suitably controlled, is used for transmission. A detailed description of the Homestake installation is given by J. F. Wiggert. Another design in which shaft wires are used is described by L. D. Stewart. (See the bibliography for references.)

Hoist Safety Controls.—Hoists are equipped with a position indicator for each drum and the corresponding shaft vehicle. Track or limit switches are installed in the shaft and headframe to prevent overwinding and near shaft bottoms to prevent lowering into the sump. Each hoist drum is also equipped with an automatic device to prevent overspeed and overwind. It is used in conjunction with the track or limit switches. The Lilly hoist control, gravity or electric, is widely used for this purpose. Many different designs are available to suit the wide range in hoisting conditions and various types of hoist. The control is provided with a ball governor, which gives warning when operating speed passes a certain limit and then shuts off power and applies the brakes if the warning is disregarded; it is also provided with a disk or wheel, on the periphery of which are cams establishing the limits of top and bottom travel and also for operating the mechanism for retardation and regulating the application of the brakes. Warning is given when to begin retarding for the top or bottom of the trip. If the warning signal is disregarded, power is shut off and the brakes are applied; likewise if the hoist is started in the wrong direction at top or bottom. If the power for an electrically operated hoist fails, the controller will immediately apply the brakes. The control has also been incorporated in the mechanism for automatic hoists. Each drum of the hoist is coupled by chain and sprockets or by gears to its hoist controller.

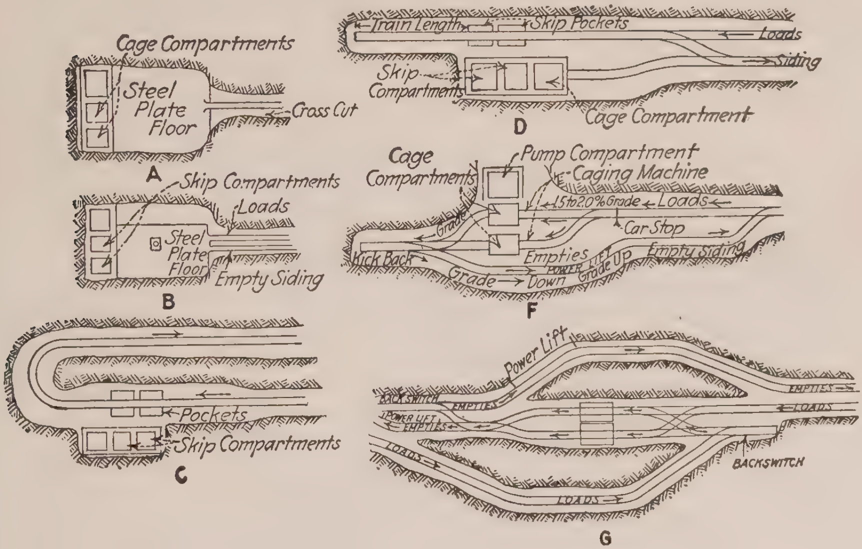


FIG. 97.—Typical shaft station plans. (A), (B), (C), and (D) are used in metal-mining practice; (F) and (G) in coal-mining practice.

Stations and Main Levels.—Station plans for metal mines fall into a few well-defined types. Necessarily the method of hoisting, whether by cage or skip, the method of haulage, the tonnage to be supplied by the station, and the nature and amount of supplies to be delivered to the station will determine the size of the station and the details of the plan. In Fig. 97, several types are shown. *A* represents a plan used it small mines and suitable for cage hoisting. *B* is suitable for skip hoisting and for handling a moderate tonnage. *C* is a plan much used where locomotive haulage is required *D* in various modifications is a plan used for main haulage levels in large mines. It provides for skip compartments, shaft loading pockets, and a large cage compartment with separate sidings for the cage; it may be readily designed for the modern square shaft. *F* and *G* represent single-level shaft stations much used in coal-mining practice where large tonnages are hoisted in cages, which may be of either level-deck or tilting-deck construction. Caging machines are used at the loading level; one is also required at the surface where level-deck cages are used.

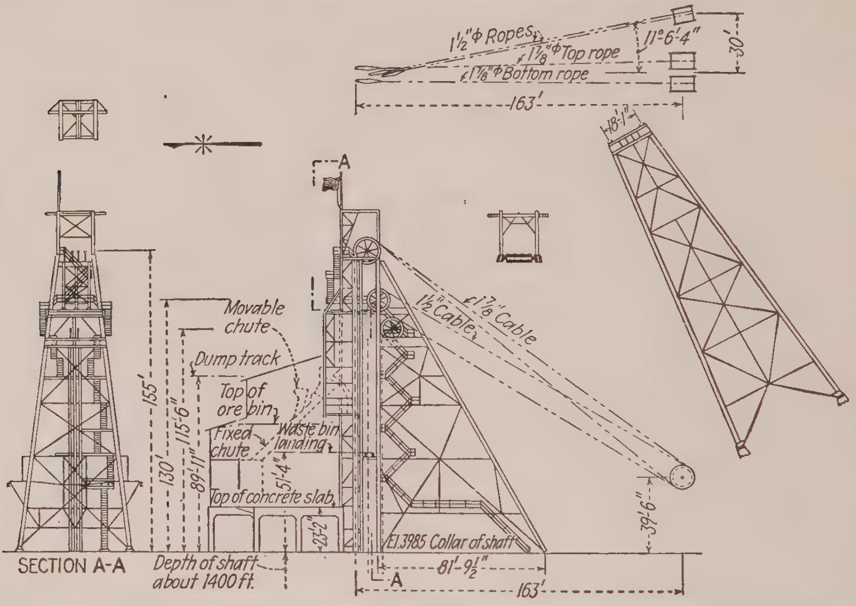


Fig. 98.—Headframe and surface bins, Porphyry shaft, Inspiration mine. (*G. H. Booth, Eng. Min. Jour.*)

Surface Arrangements.—Shafts are equipped with headframes (Fig. 98), which support the hoisting sheaves, landing platforms, and receiving bins for ore and waste rock. Mechanical arrangements for changing skips and cages may also be provided. In the simplest rig the landing stage is at the collar of the shaft; the headframe may be of timber and some 40 to 75 ft. in height. For large mines and deep shafts the headframe may be of steel, with a housing for landing decks and sheaves in severe climates. Reinforced-concrete construction is used to some extent. The height of headframes ranges from 100 to 185 ft., very high structures being required for very deep shafts in which high-speed hoisting is necessary. Steel or timber bridges connect the landing decks with waste-rock tracks and ore bins or stock piles. Figure 99 shows two surface rigs for handling buckets used for either sinking or ore hoisting in small operations.

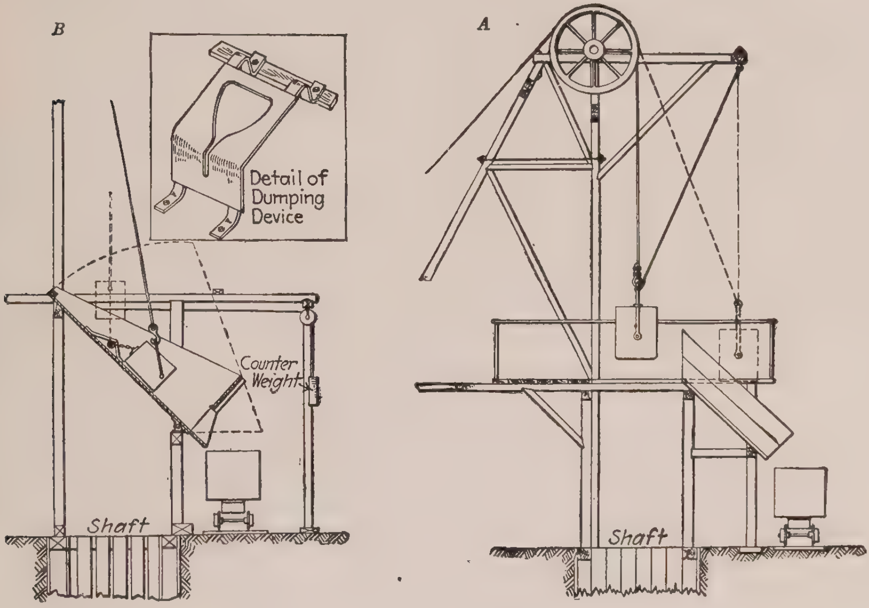


Fig. 99.—Methods for dumping buckets at shaft collar.

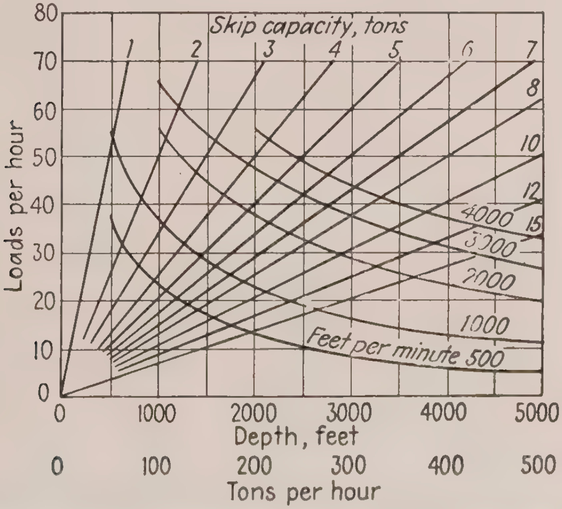


Fig. 99a.—Hourly hoisting capacity, estimated for two-compartment hoisting, based on assumption of loading time 10 sec., acceleration 15 sec., and retardation 15 sec.; for various speeds and shaft depths.

Capacity of Hoisting Installations.—Shaft capacity in ore hoisting is shift time divided by the total average time required for one cycle, which may include loading, hoisting, unloading, and lowering, into the tons per skip load. In two-compartment hoisting, one skip is lowered while the other is being hoisted; likewise one skip is being loaded while the other is emptying. Figure 99*a* is a chart for estimating shaft capacity in two-compartment hoisting.

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CHAPTER VIII

MINE DRAINAGE

GENERAL FEATURES

Occurrence of Underground Water.—Underground waters owe their source to surface waters, which penetrate porous strata and slowly move laterally and downward until they reach equilibrium, or to deep-seated springs that feed fissures or systems of fissures. Water from deep-seated sources ascends until either it flows out from some vent at the surface or equilibrium stops further upper movement. Porous or fissured rock masses may be looked upon as possible water containers at depth. Rocks such as clay shale are practically impervious and may prevent the upward flow of water, or if water is above them they may prevent the downward flow into mine workings. Many compact sedimentaries and igneous rocks, except where extensively jointed and fissured, either do not contain much water or are impervious. Alluvial material, sand, gravel, and the like are porous and permit more or less free movement of water. Limestones are often cavernous and water-bearing. From a topographic standpoint basins and valleys are apt to contain water in their lowest level at no inconsiderable depth from the surface. The slopes leading to the valleys contain water, but at increasing depth as greater elevation above the intervening valley is attained. On mountain ranges the distribution of the underground water is erratic, and sometimes unexpected quantities are encountered in mine workings. The water plane or the top surface of the mass of water below the ground surface is more or less irregular in regions of low relief and extremely irregular in regions of high relief.

In desert countries the water plane may be conspicuously absent or at such great depth as to be encountered only by the deepest workings. At Tonopah, in Nevada, water was struck in small quantities at depths of about 1,000 ft., and in the neighboring camp at Goldfield the same condition prevailed. In humid localities and in elevated regions of moderate relief and moderate rainfall, underground water accumulations are frequently large and must always be expected unless rock masses are conspicuously impervious and free from fissured zones.

A mine working above the water plane may be expected to encounter only seepage or incidental quantities, but where a working penetrates the water plane there is a general movement of the water into the excavation.

In a shaft the water rises if undisturbed until it reaches the level of the water plane. The rapidity with which it flows in depends upon the depth of the shaft bottom below the water plane and the porosity of the surrounding rock mass. As it is drained and deepened, the depth below the water plane becomes greater; the head also becomes greater and the rate of inflow is increased. This is in accordance with the principle that as the depth of drainage is increased the drainage area is proportionally enlarged. In time the water plane about the shaft is lowered and the rate of inflow diminishes. Where workings are extended from a shaft the rate of inflow increases. If we imagine the rock mass to be of uniform porosity, such an increase might be expected to be proportional to the extent of the workings. Ultimately the rock mass above would be drained and the floor of the workings would represent an axis in a valleylike water plane. At this stage the amount of inflow would be reduced to a minimum and would be represented for the most part by the accessions coming from the surface. The general observation that can be deduced is that the water inflow in a mine working may be expected to reach a maximum at or about the time the working is completed, and as time goes on the inflow decreases to a minimum. Just how long the period between maximum and minimum flow will be depends upon the drainage rate and the amount of water that must be removed before the general lowering and flattening of the water plane takes place.

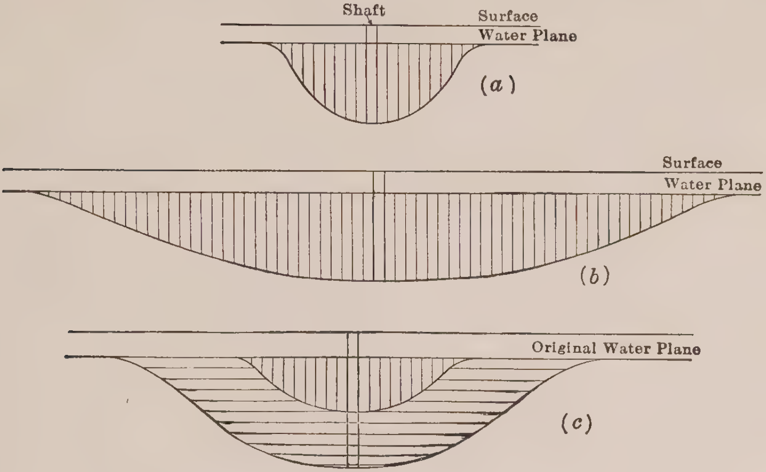


Fig. 100.—Hypothetical ground-water levels.

In Figs. 100*a*, *b*, and *c* hypothetical cases are assumed and the changes caused by shaft drainage that are made in the water plane represented. In *a* it is assumed that the rock mass is of small porosity. The volume drained by the shaft is shown by the shaded portion and the bounding line is the new water plane. In *b* the rock mass is of greater porosity and admits of freer movement of the underground water. A relatively greater area and volume are affected. In *c* the effect of deepening the shaft is illustrated.

The geological examination of an ore deposit should not neglect a careful study of the possible occurrence of underground water and of structural features that might serve as channels for the passage or zones for the accumulation of large quantities of water. The position of an ore deposit relative to topography and surface drainage should receive attention. The porosity of the rock masses, especially in the zone of surface drainage tributary to the deposit, is an important element. Mines, wells, springs, streams, and lakes in the vicinity may supply much evidence of underground water.

Occurrence of Water in Mining Districts.—Very deep mines are not often wet mines. As an example, in the Kolar district (India) where some of the mines have reached a depth of 8,500 ft., T. Pryor states that the average water pumped per mine, including workings on lodes other than the Champion, is less than 300 gal. per min. In the Butte district (Montana) M. McCanna, in describing the High Ore shaft pumping unit, indicated an over-all pump capacity of 6,000 gal. per min. with provision for an additional 3,000 gal. per min., or an over-all capacity of 9,000 gal. per min. The Frood mine (Canada) has a pumping station of 576 gal. per min. with additional reserve capacity on its 2,800-ft. level, and pumps an average of 85 gal. per min. Michigan copper mines at a depth of 6,000 ft. are said to be practically dry on lower levels; the Calumet and Hecla is said to have pumped over 2,250,000 gal. per day, the water originating in the upper workings principally. The lower half of the Morro Velho Mine (Brazil), reaching a depth of about 8,000 ft., is said to be bone-dry. The Magma mine (Arizona) has a 1,200-gal. per min. pumping station on its 3,600-ft. level. Two 600-gal. per min. units are used alternately for 16 to 20 hr. per day.

Comparatively shallow mines at high altitudes often have a water problem. The Morococha unit of the Cerro de Pasco (Peru) had at one time an installed capacity of 30,000 gal. per min. on its 1,014-ft. level of the Natividad mine, the lift being 520 ft. (A. C. MacHardy). At Jarbidge (Nevada) at 6,200 ft. altitude 7 billion gal. of water were pumped in sinking 1,100 ft. and developing 2,300 ft. of short, wet laterals; maximum pump capacity installed lifted 7,100 gal. per min. from the 1,000-ft. level to the surface (R. O. Camozzi). At Venterspost (South Africa) shaft sinking in dolomite was attempted, but at 97 ft. pumping capacity of 30,000 gal. per min. failed to meet requirements. In 1911 some 5 million gal. per day was pumped. Shaft sinking was abandoned but revived in 1934, when the shaft was successfully sunk by cementation. The

Cripple Creek district (Colorado) ranges in altitude from 9,000 to 10,100 ft. with a medium- to high-relief topography. The annual precipitation is considerable and the ground structure favors fissure water. The mines are drained by tunnels and by pumping, the amounts of water varying over a wide range. Discharge of the Carlton tunnel varied from 8,500 to 25,000 gal. per min. The amount of water pumped at Cripple Creek in 1936 is shown in Table 30*A*, and Table 31 gives data on the drain tunnels constructed at various times in the district. Figure 101 is a longitudinal section showing the vertical position of the drain tunnels.

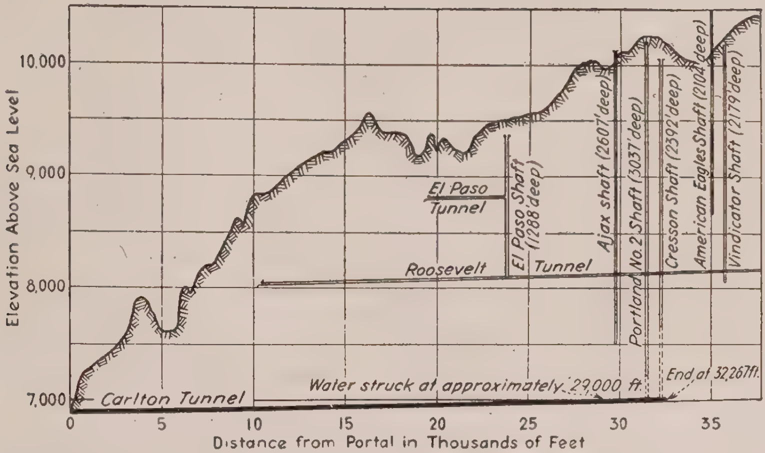


FIG. 101.—Vertical projection and section of Cripple Creek district. (*Eng. Min. Jour.*)

TABLE 30*A*.—CRIPPLE CREEK (COLORADO) PUMPING IN 1936

Mine	Type of pump	Pumping head, ft.	Capacity, g.p.m.
Ajax	Centrifugal	400	3,000
Cresson	Air-lift	127	2,100
Vindicator	Centrifugal	125	1,400

Fortunately, in most cases of excessive water the continuous flow after the initial drainage of a mining area may be only a fraction of the dewatering stage. Most high-altitude mines and those in natural drainage basins have a water problem, as do also mines in highly fissured and faulted areas subject to rainfall of ordinary proportions. Structural geology, type of rocks, topography, and precipitation need to be studied. Even under desert conditions accumulated fissure water may occur in large quantity. The Pennsylvania anthracite mines have a serious water problem, the average being estimated about 11 tons of water pumped to 1 ton of coal mined; bituminous mines in the same state average about

2 tons of water per ton of coal mined. In the Picher area of the Tri-State zinc district during a period of heavy precipitation in 1929, the estimated water pumped exceeded 10 billion gal.; the average annual pumping costs over a 5-year period, 1928–32, were about $130,000 (E. W. McCuskey).

TABLE 31.—CRIPPLE CREEK DRAIN TUNNELS

Tunnel	Date	Elevation, portal, ft.	Discharge, g.p.m.
Blue Bell	....	9,342	200– 300
Ophelia	1896	9,275	1,000– 3,000
Standard	1898	9,034	1,000–12,000
El Paso	1902	8,790	1,800– 6,800
Roosevelt[a]	1907–18	8,020	.....–16,000
Carlton[b]	1941	6,893	8,500–25,000

[a] Length, 26,000 ft.

[b] Length, 32,267 ft., grade 0.3 per cent.

Properties of Water.—Although the specific gravity of water is unity at 39.3°F. its maximum density, it is usually assumed as unity in calculations. The viscosity of water is greatest at 32°F. and becomes less as its temperature increases, but this decrease is seldom taken into consideration. The unit taken in drainage calculations is the U.S. gallon of 231 cu. in., weight 8.342 lb.; the imperial gallon containing 277.41 cu. in. and weighing 10.02 lb. is used in British mining practice.

The corrosiveness of water is important, as this property influences the selection of materials used for pumps, pipe, valves, and fittings. Analysis indicates the degree of acidity or alkalinity. Acidity is expressed in parts per million (p.p.m.) of calcium carbonate that would be necessary for neutralization; total acidity, p.p.m., represents free acidity and also that caused by iron salts in solution. The "pH" indicates the free hydrogen ions in solution. Neutral water has a value of 7; less than 7 represents an acid condition and more than 7 an alkaline condition. The numbers expressing pH are the logarithms of the reciprocals of the normality of H ions in solution. Standard methods of water analysis are used. Salts in solution are of interest as they may form incrustations in pipes. Suspensions such as sand or grit may cause abrasion and wear of pump parts. Most mine waters are acid in varying degree, especially those in sulphide and copper deposits. The principal incrusting salts are lime and iron compounds. An analysis of mine water is necessary, as well as the determination of the amount and kind of suspended matter. The desirability of conditioning water by settlement to remove grit, or by neutralization to reduce free acid and to precipitate iron compounds and copper, is often considered.

Examples of acid waters from several coal mines in western Pennsylvania are as follows: In parts per million, free acidity due to free sulphuric acid ranged from 270 to 3,200 with averages from 430 to 2,160; total acidity due to free acid and sulphates ranged from 2,180 to 17,200 with averages from 2,850 to 14,160. R. D. Leitch analyzed outflowing waters from a number of western Pennsylvania coal mines, and his results showed ranges in free acidity from alkaline up to 299 with total acidities reaching as high as 14,484 and pH of the acid samples less than 3.2. H. L. Lavender gives examples of mine waters in the Bisbee area (Arizona), where flows ranged from 98 to 452 gal. per min., free acidity 31 parts per million, and copper content from 68 to 1,800 parts per million.

Lime has been used for the neutralizing of acid waters before delivery to the pump sumps. It is mixed in an agitating tank and the slurry is added in a small stream to the water as it flows to the sumps. An excess of lime is usually required, with the result that lime compounds accumulate and ditches and sumps must necessarily be cleaned at frequent intervals. H. L. Lavender described a method for handling acid copper-bearing mine water that replaced lime neutralization in some of the Bisbee mines. The mine water was passed through a series of waterproofed concrete boxes, each 10 ft. long by 5 ft. wide and 5 ft. high, divided into two compartments. Sloping bottoms and a side door to each compartment gave access for cleaning; a wooden grid was placed about a foot above the bottom. The boxes were filled with scrap iron. Perforated lead pipes distributed air at 3-lb. pressure beneath the grid. A small Root blower supplied the air-header pipe. The scrap iron precipitated the copper, neutralized the acidity, and also reduced the ferric salts in solution to a ferrous condition. Most of the copper was precipitated in the first three boxes. The principal labor required was for loading the boxes with scrap iron and for cleaning out the precipitated copper.

Discharge of acid mine waters into small streams results in pollution that may be highly objectionable. Neutralization by finely divided limestone with subsequent settlement is probably the only effective method to control stream pollution caused by acid mine water. In some cases a separate plant may be necessary for this purpose.

Corrosion-resistant metals for acid waters are available and are used in pump construction. L. Weaver lists the following as highly resistant: High-chrome steel and iron, high-silicon iron (silicon 14 per cent or over), lead-base alloys. Less highly resistant but used to a great extent are lead bronzes, bronze not containing zinc, and high-nickel copper alloys. Metals not suitable are gray iron, malleable iron, cast steel, brass or bronze containing zinc, and carbon and alloy steels other than high-chrome steel.

Economics of Mine Drainage.—The removal of water from a mine entails expense. Much water pumped against a high head greatly increases the mining cost. The flooding of a mine as the result of heavy storms or unanticipated water entrance may cause a mine to be abandoned. The capital cost of the drainage equipment or facilities plus its operating and maintenance expense during the productive years of a mine divided by the tonnage produced should result in a low unit cost, or should not be prohibitive when compared to the ore value per ton. The opening of a mineral deposit is an adventure attended by many uncertain quantities, among which is sometimes the question of drainage. Its cost may be of major importance, though it is rarely a prohibitive factor.

Where pumping may be necessary the capacity of the plant is great enough to handle the maximum flow of water that may be expected, but the interval of time required for this service is usually only a fraction of the total time of pumping. Capital input is thus much greater where peak flows are characteristic of the mine. Most mines have a peak flow period, and at such times greater attendance may be necessary. Maintenance is also much greater where pumps are driven hard.

Power is usually the greatest cost item. The design and selection of high-efficiency pumping units, as well as a pipe system in which friction heads are a minimum, are of great importance where large quantities of water must be removed from a mine over a number of years. Where high heads must be overcome the importance of these requirements is enhanced.

Concentration of pumping units reduces attendance labor and facilitates maintenance work. Small, scattered pumping units require a much greater proportion of labor relatively than do the centralized drainage plants.

Depreciation, cost of replacements, repairs, and maintenance are properly added to direct operating costs. Well-designed and -installed drainage plants usually show diminished replacement, repair, and maintenance costs. Consideration of power rates, peak loads, and other operating power loads may result in more even power demand with lower power costs.

Engineering Data of Mine Drainage.—In the initial operations at a mine the specific data required for the solution of the drainage problem are not often obtainable, but as the development progresses careful observation may supply sufficient information to anticipate the major difficulties. The preliminary work which has been already suggested and which is largely of a geological nature can, however, be carried out. To this may be added the meteorological data, the average annual rainfall, the seasonal rainfall, and the proportion of runoff. Topographic observations can also be made. These would consist of the determination of the drainage area tributary to the ore deposit and the situation of the deposit in its relation to the possible drainage of the workings by a tunnel.

Work of this kind is reconnaissance work and requires the determination by aneroid of the elevations of the deeper topographic depressions within a radius of from 2 to 4 miles of the deposit. Where the relief is low a drain tunnel is obviously out of the question and the studies suggested are unnecessary, but with high relief much of value may be obtained by a general reconnaissance. The experience of neighboring mines, if there are any, should be carefully investigated. The specific questions for which an answer should be sought are: What is the annual inflow into the mine? At what time does the maximum rate of inflow occur and what is its duration? Is there danger of the penetration of zones containing large bodies of water, and if so, what has been the experience of neighboring mines with respect to the quantity of water removed from such zones? Are neighboring mines in such a position as to reduce materially the amount of water required to be handled in the mine in question? Can careful mining avoid water zones or can advantage be taken of impervious or watertight formations? Can surface or underground water be sealed off or prevented from entering the mine? Does the water occur in the upper part of a mine or must water be handled on each level? Is the ore zone a drainage channel? What are the limiting topographical conditions that would determine the height to which the water must be raised? Are the limiting topographic conditions such as to indicate the probability of a drain tunnel? What would be the practical difficulties of driving such a tunnel? What would be the cost and time required? What is the chemical nature of the water, *i.e.*, does it contain acid or incrustating substances?

Rarely is the drainage problem capable of solution at the beginning of operations, but it keeps step with the development and from time to time comes up for general revision during the life of a mine. Temporary appliances are supplanted by more permanent and efficient appliances and methods. Individual mines cooperate with other mines in the near neighborhood when the drainage problem becomes too large for a single mine to handle. Thus collective action may result in a more equitable division of the burden and may serve to prolong the life of an entire mining district that faces extinction by the encroachment of excessive quantities of water.

METHODS OF DRAINAGE

Drain Tunnels and Ditches.—The simplest method of drainage is by a tunnel so placed as to intersect mine workings below the lowest stratum of the ore bodies or, where the topography is such as to make this impracticable, at some intermediate level that will provide drainage for the higher ore bodies and reduce the pumping head for the lower ore bodies. The length and cost of the drain tunnel as well as the vertical drainage expectation are the important factors. Such tunnels may cost from $10 to $30 per foot and extend in length up to 5 or 6 miles. They may also serve for development and haulage. Table 32 gives examples of drain tunnels.

TABLE 32.—DRAIN TUNNELS

Name	Location	Purpose	Date	Cross section		Length, ft.
				Width, ft.	Height, ft.	
Sutro	Nev.	D and T	1879	11	8	20,489
Burleigh	Colo.	D and T	1868	6	7	3,000
Carter	"	D and T	1897	5½	7½	7,600
Central	"	D and T	....	12	8	
				5	7	9,000
Lausanne	Pa.	D	....	12	8	20,000
Newhouse	Colo.	D and T	1893	8	8	22,000
Ontario	Utah	D	1888	5	7½	24,000
Rawley	Colo.	D and D	1911	8	7	6,235
Roosevelt	"	D	1907	10	6	26,000
Yak	"	D and T	1895	7	7	23,800
Carlton	"	D and T	1941	10	11	32,267
Elton	Utah	D and T	....	11	10	24,000
Kingsmill	Peru	D and T	1943	9	12	32,000
Leadville	Colo.	D	Proposed	9½	10	13,500

NOTE.—D = drainage; D and T, drainage and transportation; D and D, drainage and development.

Excessive amounts of water indicate a tunnel where topography is favorable, but water flows of 500 to 1,000 gal. per min. may be and are frequently handled more economically by pumping. Time is also a factor. To drive a tunnel of several miles in length may delay the development of a mine for a considerable period. To pump 1,000 gal. per min. against a 100-ft. head requires 565.7 kw.-hr. per day at 80 per cent efficiency. At 1 cent per kw.-hr. the power cost is $5.65 per day per 100 ft. of head. Every effort is therefore made to reduce pumping heads to limits imposed by topographical environment. Thus the 32,000-ft. Carlton tunnel in the Cripple Creek district, which was constructed to gain over 1,000 ft. of vertical drainage below an upper drain tunnel, effected an important economy in mining. In 1928, Cerro de Pasco started a 9- by 12-ft. tunnel (Mahr-Kingsmill) to drain its Morococha mines, completing it to 32,000 ft. to gain 700 ft. below the 1,000-ft. level. The pumping station at this level had an installed capacity of 30,000 gal. per min. against a head of about 400 ft., but the maximum capacity pumped for any length of time was 18,000 gal. per min. It was dismantled on completion of the tunnel.

Surface drainage is provided at shaft collars by drain ditches so placed as to take surface runoff away from the shaft mouth. Shafts in a flat or basin subject to overflow are protected by elevating the shaft collars above the anticipated water levels and sealing the upper part of the shaft from water infiltration.

Open pits are drained by ditches where the topography permits. Drain tunnels may be necessary, or an unfavorable topography may require the installation of a pumping plant. Wherever possible, surface water is ditched away from the pit limits.

By-passing and Sealing Off Water.—Heavy flows of water encountered in tunnel driving are sometimes handled by deflecting the course of the tunnel around the wet area. Another method is to seal off the water by injecting cement grout through boreholes driven from the tunnel at various angles to intersection with the watercourses. The objective is to surround the tunnel section with an impervious cylinder within which the watercourses are filled with cement grout. A similar plan is followed in shaft sinking where uncontrollable amounts of water are encountered.

Where known water zones neighbor mining areas, development is planned to avoid as far as may be practicable the intersecting of such areas. Where it is anticipated that intersection may be expected or is unavoidable, crosscuts are provided with bulkheads and pressure-resisting doors to control heavy water flows. Diamond-drill holes are drilled in advance of workings to determine the position of the watercourses.

W. W. Weigel describes the practice in the southeast Missouri lead district of sealing water-charged fissures by slime-grouting through diamond-drill holes originally bored for prospecting purposes. Supplementary holes were also drilled across the strike of known channels. Flotation tailing used for this purpose was conditioned to contain 20 per cent solids, 8 per cent on 100 mesh, and handled in pipes at a velocity of 150 ft. per min. or more. Channel zones were indicated from the diamond-drill records and by surface and geological examination. Up to the beginning of 1943 grouting had been run into 763 holes over an area of 240 acres; 360,000 tons of solids were pumped. Weigel states that there has been no evidence of any surface water leaking into any slimed area after treatment.

Many stream beds passing over mining ground are grouted where examination shows fractured areas. Other stream flows are flumed across, or the stream is deflected around the limits of the mining ground. Prevention of inflow is an important method of control and may lessen the quantity of water to be handled. A study of geological structure helps in the solution of the drainage of a mine.

Bailing.—Small water flows encountered in prospecting and exploration work are removed by bailing. The water is allowed to accumulate in the shaft sump, and part of the hoisting time is devoted to water removal. Bailing was at one time an accepted method for draining large mines, and shafts were equipped for this special purpose. It is now unimportant, its principal use being an auxiliary in dewatering flooded mines.

Pumping.—Water occurring below the gravity drainage is collected in sumps and removed by pumping. Drainage points are distributed at advantageous points upon a level, and pumping stations are placed near the shafts. Auxiliary pumps discharge into gravity drainage leading to the collecting sumps. Ditches and pipes are used. In shafts water rings collect the water, and pipes conduct it to the nearest sump. Figure 102 shows details of water-ring construction.

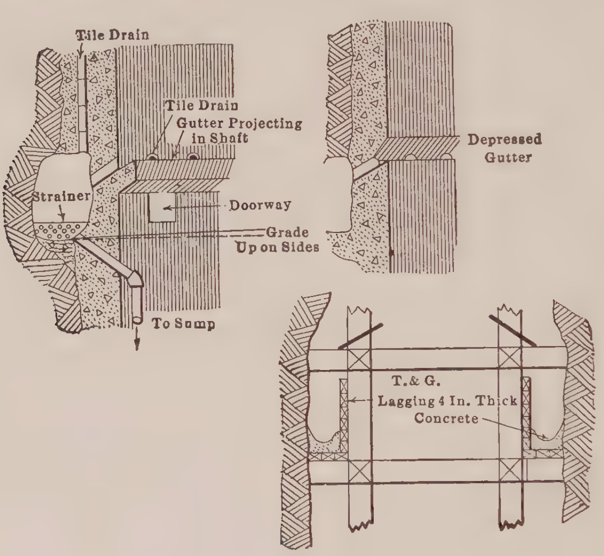


Fig. 102.—Water-rings in shafts.

In deep mines water is pumped in stages ranging from 500 to 2,000 ft. per stage, and in very deep mines lifts of from 2,000 to 3,000 ft. are not uncommon. At the Magma mine (Arizona) a single lift against a 3,304-ft. head is employed. High pumping heads involve high pressures, corresponding to which are heavy pumps, pipes, valves, and fittings. With a lower lift several pump stations in series may be necessary, each of which will require lighter equipment than the single-lift unit, but the aggregate cost may exceed the cost of the single-lift station with its heavy equipment. The single-lift station requires less attendance than the series lift. However, automatic pump control enables a series lift to be handled with about the same labor as a single-lift installation. Each deep mine is a separate problem in engineering and design.

Capacity is obtained by one or more units at a given station. As water inflow varies between a periodical maximum and minimum, the unit size may correspond to the minimum flow, other units being added to take care of the maximum and emergency flows. Usually all units are of the same size. Pumps range in capacity up to 1,500 to 2,500 gal. per min.; larger units up to 4,500 to 6,000 gal. per min. may be employed where heads are about 500 ft. or less. Single-stage pumping is general in mines 500 to 1,000 ft. in depth. In some pump stations small pumps are installed to care for minimum and average flows, and one or more larger units are installed to handle maximum and emergency flows. Table 33 gives examples of pumping plants required in a large lead mine.

TABLE 33.—PUMPING EQUIPMENT, 1942, SOUTHEAST MISSOURI LEAD DISTRICT

Division	Stations	Shaft pumps	Pumps, hp.	Total capacity, g.p.m.	Relay pumps, stations	Pumps	Hp.
Federal	4	12	3,800	24,600	10	20	1,175
Leadwood	7	30	3,350	18,680	13	21	312
Desloge	2	7	965	5,000	8	10	130
Bonne Terre	2	6	875	3,600	10	13	115
Total	15	45	8,090	51,880	41	64	1,732

NOTE.—All shaft pumps are centrifugal.

Mines handle 20,000 gal. per min. against 465-ft. head for daily tonnage of 21,200, or 6½ tons of water per ton of ore. (W. W. Weigel, *T.P.* 1607, *A.I.M.E.*)

Electric power has superseded steam and compressed air, although compressed air is still used for sinking pumps and for small auxiliary pumps in many instances. Alternating current of 2,200 volts or more is delivered to pumping stations through armored shaft cables to control panels and thence to the motors. In some cases transformers are installed underground and higher voltages are used in the shaft transmission cables. Small pumps are operated at lower voltages. Induction motors are usual, but some synchronous motors are used with large units.

Four types of service may be provided—sinking, station, auxiliary, and dewatering of flooded mines. Specialized pumps are required during shaft sinking where water is encountered. Station service handles water inflow during mine operation. Auxiliary pumps remove water from low places in development workings and deliver to gravity drainage. Dewatering requires special pumping equipment and methods. After dewatering, during which a large volume of water may be removed in a relatively short period, the special equipment is removed and the normal and seasonal inflow is handled by permanently installed equipment.

A drainage unit consists of a sump, settling basins discharging to the sump, and the pump or pumps with suction and column pipes, control valves, motors, and control panels. The unit is placed close to the shaft in which are installed the column pipes extending to the surface or to the drain tunnel. Where highly acid or copper-bearing water is handled, tanks are installed for neutralizing and for precipitating copper before the water is discharged into the sump. Where muddy water requires clarification and grit must be removed, special settling sumps are provided. Additional equipment is usually necessary for the removal and disposal of the sludge accumulating in these special settling sumps.

MECHANICS OF PUMPING

Work.—The mechanical work of elevating a given quantity of water a given vertical distance is readily computed by taking the product of the weight of water per minute and the vertical distance. This gives the actual work in foot-pounds per minute. The total work or input which must be supplied to the prime mover is the sum of the actual work, the work of friction in the suction and column pipes and fittings, the work of friction in the pump and prime mover, and the work necessary to accelerate the given mass of water from rest to the maximum velocity of flow. The work of friction in the pipes and fittings is the product of the friction head and the weight of water per minute. Pipe friction depends on the length of the pipe and the velocity of flow. It is expressed in terms of feet of head, and tables containing its value are to be found in engineering handbooks. The frictional resistance caused by fittings and valves is also expressed in terms of feet of head or sometimes in length of straight pipe having an equivalent frictional resistance. The work is, as before, the product of weight and head. The work of friction in the pump is occasioned by the friction of the bearings, moving parts, and packing glands and the movement of the water through the pump. Where gears or silent-chain drives are used the work lost in the transmission of the power is added to the foregoing. While it would be practicable to figure out each loss, it is more convenient to group all losses together and express them as loss in efficiency. With the prime mover a similar procedure is followed.

Total head (H) may include the static head or vertical distance between water surfaces of sump and discharge, friction head of pipe, valves, and fittings (obtained from tables) in feet, and the velocity head, which is velocity in feet per second divided by 2g. Acceleration of gravity, g, may be taken at approximately 32 ft. per sec. The weight of the water lifted per minute multiplied by the total head in feet gives the work performed in foot-pounds per minute. Dividing by 33,000 gives the horsepower theoretically required. Multiplying by 100 and dividing by the efficiency factor in per cent will give the total estimated horsepower. Figure 103 gives the theoretical power requirements for pumping 1,000 gal. per min. against a static head of 1,000 ft. under several conditions.

Efficiency.—Efficiency is the ratio of useful work to input work expressed as a percentage. Over-all efficiency is defined in various ways and may take into account the power losses in each conversion from the source of power to the delivery of the water to gravity drainage. Usually over-all efficiency is figured upon the basis of the power delivered to the pothead at the shaft, the transmission losses to the pump motor, the power loss of the motor, the power loss of the pump, and the power required to overcome friction in pipes, valves, and fittings. Plunger pumps have a mechanical efficiency of 90 to 95 per cent, turbine pumps 75 to 90 per cent, gear drives as high as 97 per cent, motors 90 per cent, transmission lines 97 per cent, and transformers 98 per cent. Over-all efficiency thus includes all the successive power losses and friction to be overcome. Over-all efficiencies range from 50 to 70 per cent.

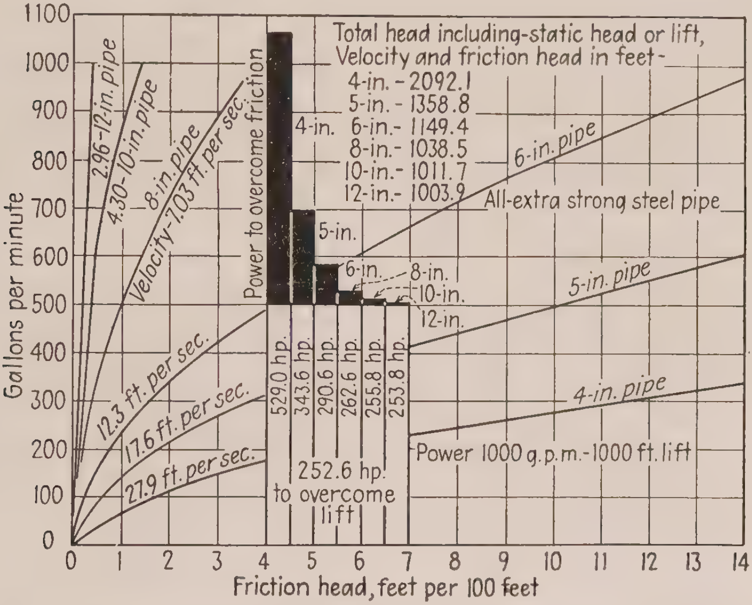


FIG. 103.—Influence of pipe friction on power required in pumping; 4-, 5-, 6-, 8-, 10-, and 12-in. pipe is compared on basis of 1,000 gal. per min. and 1,000 ft. static head.

Pumps are operated at rated capacities for maximum efficiency. A pumping station operating at full capacity may be expected to give a high efficiency where mechanical and electrical design are good, but at partial capacity the efficiency may be considerably less.

The capacity of a plunger pump is from 90 to 98 per cent of its plunger displacement, the loss in efficiency being due to the time element in the closing of the valves and to some leakage at the packing glands. Poor maintenance may reduce this capacity greatly. Centrifugal and turbine pumps likewise have slip in variable amounts, depending on design, wear, and head. The capacities of such pumps are given for various heads and operating speeds by the manufacturers.

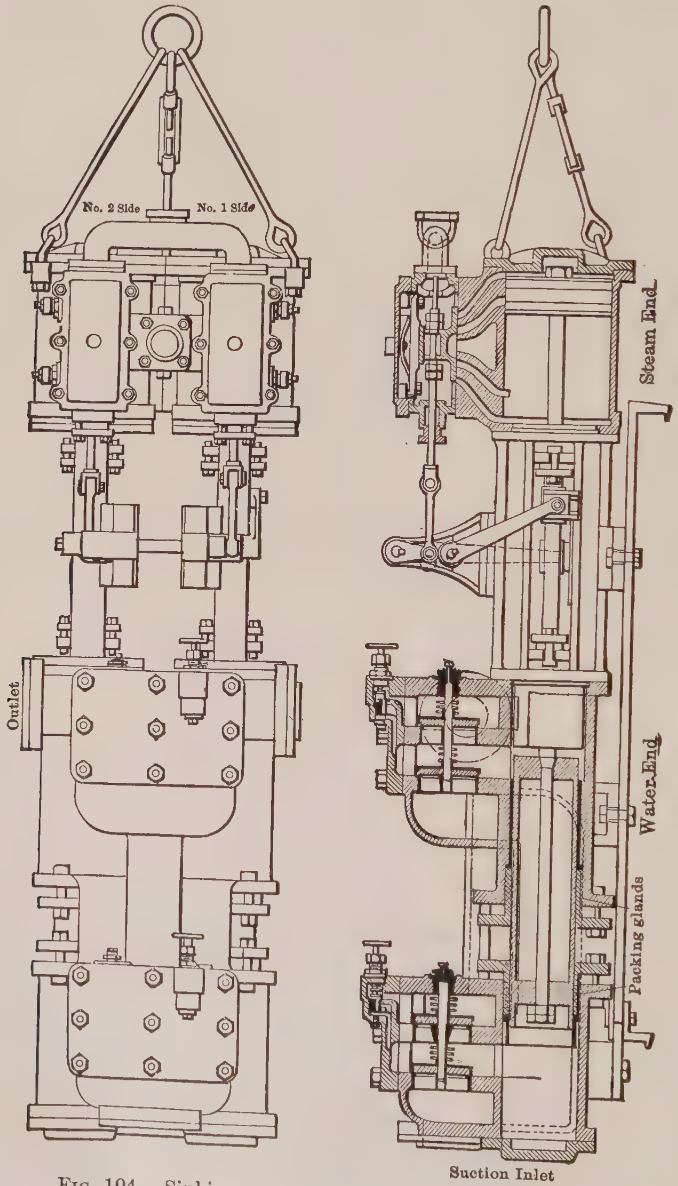


FIG. 104.—Sinking pump. (*Prescott Steam Pump Co.*)

TYPES OF MINE PUMPS

Plunger Pumps.—Plunger or reciprocating pumps include sinkers for shaft work and vertical multiplunger and horizontal single or duplex units for station service. The sinking pump (Fig. 104) is a double-acting outside-packed plunger, driven by a compressed-air cylinder in line with the water end. Single and duplex units, designed for capacities of 50 to 1,000 gal. per min. and heads from 200 to 400 ft., are obtainable. The pump is compact and is supported by hangers from the shaft timbers. It is raised and lowered by chain blocks. Pump valves are flat rubber disks seated by spiral springs. A telescopic column pipe reduces the frequency of column-pipe additions, as the pump must be kept within suction distance of the shaft bottom. The suction pipe is a reinforced rubber hose with a foot valve and basket at its inlet. Sinking pumps are subjected to the severest service, so that reserve pumps are necessary. A motor-driven multiplunger pump designed for sinking service is also in use. Vertical motor-driven centrifugals of the sinker type (Fig. 105) have superseded plunger-type sinkers to a considerable extent.

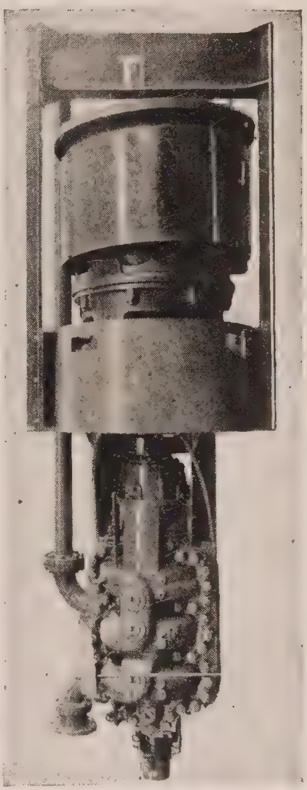


FIG. 105.—Vertical shaft pump. (*Byron Jackson Co.*)

The multiplunger pump consists of three or five single-acting plungers driven from a single crankshaft by motor and gears. It is arranged as a compact vertical unit with suction and outlet valves and suction and outlet pipes, with the motor carried upon an extension of the base plate. In one design the motor is placed above the crankshaft, providing a safe position in the event of flooding of the station. At one time belt-driven pumps of this type, arranged horizontally, were used. Pumping heads of multiplunger pumps range from 400 to 1,000 ft., plunger speeds from 80 to 100 ft. per min., and capacities up to 600 gal. per min. Formerly pumps of this type were widely used for station service.

The horizontal duplex double-acting plunger pump driven by a gear between the two halves of the pump, with pinion, shaft, and motor compactly placed at the end, is used where high heads, continuous service, and high efficiencies are required. The high-efficiency multistage centrifugal competes with this type and has the advantage of greater compactness and much less weight for equivalent pump capacity. Plunger speeds are from 200 to 400 ft. per min.; valve chambers contain relatively small-diameter inlet and outlet valves where high pressures prevail. Air chambers, suction chambers, and alleviators are used. At Magma (Arizona) pumps of this type are of 600 gal. per min. capacity, designed for a 3,465-ft. head (1,500 lb. per sq. in.); they have 6¼- by 24-in. center-packed plungers, driven through herringbone gears at 48.5 r.p.m. by a 600-hp. a-c, 3-phase, 25-cycle, 485-r.p.m., 2,200-volt motor with a flexible coupling interposed between the motor and the pinion drive. At Ajo (Arizona) similar Prescott pumps are used for water-supply service against a head of 1,176 ft. (710 lb. per sq. in.). These have 5½- by 24-in. center-packed plungers, a speed of 100 r.p.m., a 500-hp. synchronous motor, and valves of the pot-form type. The dimensions of the Ajo pump are as follows: Width 13 ft. 10½ in., length 35 ft. 6 in., and height 12 ft. 3 in.

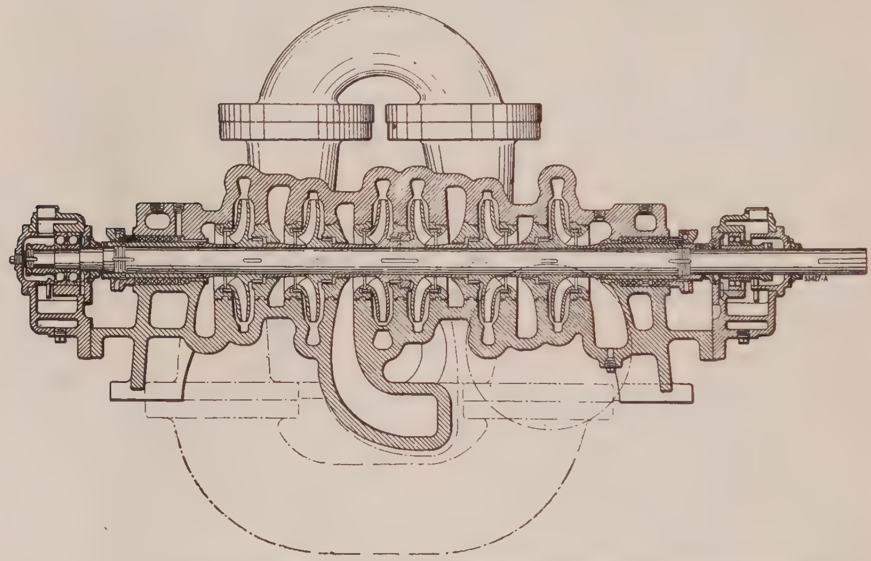


Fig. 106.—Longitudinal section of six-stage centrifugal pump. (*Ingersoll-Rand Co.*)

Centrifugal and Turbine Pumps.—The centrifugal pump consists of an open or enclosed runner surrounded by a casing with a tangential discharge outlet (volute) and a water inlet at the center. The turbine pump, in principle the reverse of the water turbine, has a shrouded impeller receiving the water at its center. A diffusion ring containing vanes surrounds the impeller and directs the impeller discharge into a circular casing, which discharges into the eye of the next impeller in series. The diffusion ring converts the high-velocity discharge of the impeller into pressure head. A crude two-stage vertical centrifugal with vanes between stages, the forerunner of the modern stage turbine pumps, appeared in 1884. The modern unit consists of two or more stages with the impellers mounted upon a common shaft. The terms "centrifugal" and "turbine" are used more or less loosely but the former refers to the volute and the latter to the diffusion vane. High speed is characteristic of all centrifugals.

TABLE 34.—MOTOR PUMPS, CENTRIFUGAL
(*Ingersoll-Rand*)

G.p.m.	Head, ft.	Hp.	Discharge, in.	Suction, in.	Dimensions, in.			Weight, lb.
					Width	Length	Height	
125	100	5	1.5	2	9⅝	26	11½	250
100	250	15	1.5	2.5	12⅝	31¾	14½	475
200	225	20	2	3	14⅞	33⅞	15¾	550
550	100	20	3	4	17¼	36⅜	18½	775
50	325	10	1.5	2.5	15	37⅜	16¼	525
100	425	20	1.5	2.5	15	40⅝	17¼	675
225	250	20	2	3	16	40⅜	18⅞	700
175	425	30	2	3	16	45⅞	19⅞	900
200	500	40	2	3	16	47¼	20⅞	1,025

NOTE.—Pumps above the line are single-stage; those below are two-stage.

TABLE 35.—H.M.T. PUMPS, DIFFUSER TYPE
(*Ingersoll-Rand*)

No. stages	Suction, in.	Discharge, in.	Width, in.	Length, in.	Height, in.	Motor length, in.
5	5	4	29½	74⅞	32½	45¾
3	6	4	34	71	44½	53½
3	8	5	38	74½	48	54
3	10	6	44	88	51½	58
3	12	8	52	97	61¾	62½

Pumps of this type develop thrust along the axis of the driving shaft. In the single-stage centrifugal, thrust is balanced by taking water on both sides of the impeller; in the multistage pump, thrust is absorbed by special thrust bearings at one end of the pump or by dividing the unit into two parts. In the latter method the stages on one side balance the thrust of the stages on the other side, and a crossover between the parts carries the the pressure water from one part to the inlet of the other part, the outlet of the latter connecting with the discharge pipe. Figure 106 shows a six-stage pump.

Pump bearings are of either the ball-bearing type or the self-aligning sleeve type. Lubrication is by grease or oil, depending on the design of the bearing. One or two stuffing boxes are necessary. Usually these are water-lubricated, receiving their water supply from the pump or from an outside source. Wearing rings, one attached to the impeller and one to the casing, take up the wear and provide a seal between the stages, the clearance between the rings determining the effectiveness of the seal. A split type of stuffing box facilitates repacking.

Single, double, and multistage centrifugals are used widely for mine-drainage service. Compact motor-pump units of the single-stage design are manufactured in capacities up to 500 gal. per min. and heads up to 200 ft.; two-stage units have capacities up to 200 gal. per min. and heads up to 500 ft. Multistage pumps are used for higher heads and capacities up to 1,500 to 3,000 gal. per min. Tables 34 and 35 give dimensions. Pumps have been constructed for heads up to 3,000 ft. Most of the larger multistage units are constructed in two parts with the motor between and a crossover between the parts. Vertical single- and two-stage pumps are frequently arranged in a steel frame with vertical motor drive and are used for shaft sinking. A special design of motor, leak-proof and protected by drip shields, is required for this service.

The deep-well type of turbine is designed for vertical operation. It consists of a series of impellers mounted upon a vertical shaft with individual bowls or casings attached to the lower end of the discharge pipe, within which the vertical shaft extends to the discharge point, where the motor drive is placed. The vertical shaft is supported at intervals of 5 to 10 ft. by bearings supported by spiders. The column pipe is in 10-ft. sections joined by screw couplings. Bearings are rubber-lined and lubricated by the water or are of the oil-lubricated sleeve type, in which case an inner pipe surrounds the shaft and its bearings. The

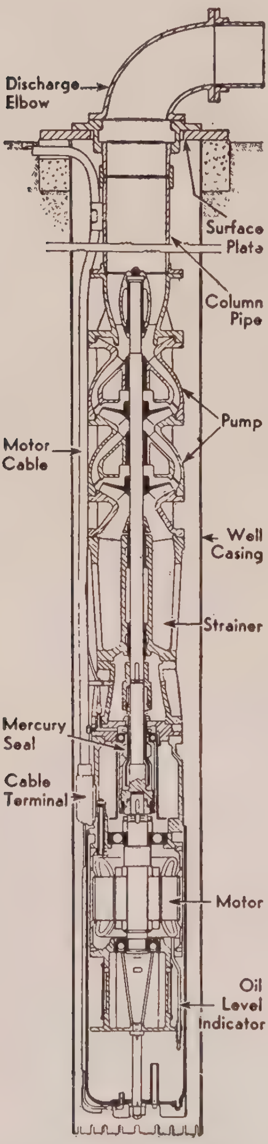


FIG. 107.—Vertical section of Byron Jackson submersible pump. (*A.I.M.E.*)

motor is built into the discharge end. The pump operates submerged. A close-coupled deep-well turbine consists of the bowl assembly attached to a short length of pipe that extends below the discharge head. The discharge of the pump is connected to a column pipe. Commercial sizes are 6, 8, 10, 12, and up to 20 in. or more, this dimension referring to the limiting well diameters within which the pump can be used. The maximum head against which the pump will operate is determined by the number of stages or bowls and by the impeller design. Large pumps of this type will give a lift ranging from 80 to 140 ft. per stage with closed impellers. Maximum head is about 800 ft. Table 36 gives sizes and capacities.

The "submersible" deep-well turbine (Fig. 107) is a deep-well unit, the shaft of which is driven by a motor below the intake of the pump. The pumping unit with its motor is attached to the column pipe. Both motor and pump are submerged. A power cable of the submarine type extends along the column pipe to the motor. The long vertical shaft is avoided. A mercury seal protects the motor from water entrance. The deep-well turbine of the shaft type appeared in 1901, and the Byron Jackson Company introduced the submersible in 1940. Both types of pumps have been used for unwatering service and have advantages over the air lift. Their efficiency is as high as that of the ordinary centrifugal stage pump. The short-coupled deep-well turbine has been used for station service but requires plenty of headroom for handling.

TABLE 36.—TYPE K DEEP-WELL TURBINE PUMP
(*Byron Jackson*)

Number	Outside dia. bowl, in.	Capacity range, g.p.m.	Minimum dia. well, in.	Preferred dia. well, in.
0K	7¼	125–500	8	10
1K	9¼	150–750	10	12
2K	11¼	250–1,350	12	14
3K	13¼	400–2,200	14	16
5K	15¼	500–3,300	16	18

Small sump pumps (sponges) of the centrifugal type, designed for operation by a compressed-air vane motor, are applicable for shaft sinking. Capacities are from 10 to 200 gal. per min. against heads of 10 to 250 ft. The pumping unit, which is self-contained, rests upon the shaft bottom and is provided with a strainer. It will drain a sump to within 3 in. of the bottom. Two units are sometimes coupled together, making a two-stage unit. An alternate drive is an electric motor. Such pumps are designed to handle sand and coarse particles up to ½ in. in diameter, discharging into a sump where settlement removes this material before the water goes to the pump sumps.

Air-lift Pump.—The air lift is the simplest form of pump. It consists of a foot piece, a column pipe within which is an air pipe, and a discharge end. The compressed-air pipe extends into the foot piece and is perforated over a short length near the lower end. The column pipe usually consists of successive sections of pipe of increasing diameter from the bottom, being largest at the discharge end. A considerable submergence is necessary for operation. The compressed air divides and lightens the water column, resulting in an upward flow of water mixed with air. The pump has a low efficiency, which increases with an increase of the submergence relative to the lift. Lift is the vertical distance between discharge and water surface; submergence is the depth of the foot piece below the water surface; per cent submergence is the ratio of submergence to the sum of lift and submergence times 100. A submergence of 25 per cent gives a lift 3 times the submergence and is close to the practicable limit; a submergence of 60 per cent may give an efficiency of 40 per cent. To start the pump, the air pressure must be sufficient to overcome the head of water within the column. Relatively large capacity, simplicity, and low maintenance are the advantages of this pump. It has been used in partially unwatering mines where the shaft is clear and the necessary submergence can be obtained, but other pumps are necessary to complete the job. It is best suited to a vertical position, though, as in the unwatering of the Idaho-Maryland mine, it can be used on inclines from 70 to 62.5 deg. In the *Transactions* of the A.I.M.E., vol. 63, p. 421, S. F. Shaw gives considerable information about the use of air lifts and in a later article (1943) the same author gives the results of tests made under different conditions. Table 37 summarizes some of these tests. The air lift has at least two possibilities, one for partially unwatering flooded mines where access is possible through a shaft and the other by the use of deep boreholes as an aid to the unwatering of formations in advance of

TABLE 37.—TESTS AT MAXIMUM CAPACITY
(*S. F. Shaw*)

Eductor pipe, in.	Lift, ft.	Submergence, ft.	Per cent submerg.	G.p.m.	Cu. ft. air per gal.	Efficiency, per cent
10	431	188	30.4	1,233	2.825	28.5
6–6⅜	492	243	33.0	358	2.40	38.7
6–6⅜	657	214	24.6	219	3.90	33.1
8½–12½	548	230	29.6	1,100	2.38	44.2
8½–12½	839	192	18.6	545	4.71	36.5
7–10	846	214	19.9	425	4.47	40.0
7–10	1,001	89	7.9	153	14.9	20.0
6–10	1,097	225	17.0	236	6.84	31.2

development. In conjunction with other methods of pumping, the air lift may be an important aid in emergencies.

PUMP STATIONS AND ACCESSORIES

The Sump.—A sump is a catch basin that receives the gravity drainage and supplies the pump inlets. It also serves as a settling basin to remove sand and sludge from the water before it is delivered to the pumps. Screens and gratings may be interposed between drainage ditches and sumps to remove trash and wood from the water. To reduce pump maintenance to a minimum, clean water is necessary, and for this reason the design of a sump (Fig. 108) should provide a sufficient settling capacity with baffles to still and break up currents as much as possible.

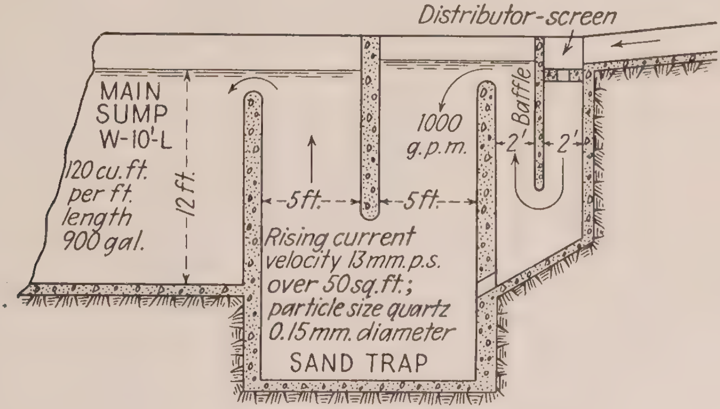

Distributor-screen
MAIN SUMP W-10'-L
120 cu. ft. per ft. length 900 gal.
12 ft.
1000 g.p.m.
Baffle
2'
2'
5 ft.
5 ft.
Rising current velocity 13mm. p.s. over 50 sq. ft.; particle size quartz 0.15mm. diameter
SAND TRAP

FIG. 108.—Sectional view of sand trap and sump.

One design provides for fluming the water from the screens to several vertical pipes, which discharge at a depth below the surface of the settlement basin equal to one-half the depth of the sump; the overflow of the basin is to the pump sump. In another, the flow from the screens is received into a distributing box across the width of the settling sump. This box discharges over a lip into the sump and against a baffle board, which directs the incoming water toward the bottom of the basin and prevents surface currents; water overflows into a series of narrow launders that deliver to a collecting launder, the narrow launders skimming off the surface water along their length. By distributing the drawoff uniformly over the surface and reducing the surface currents to a minimum, an effective separation can be obtained. In one South African sump design, six settling basins, each having an area of 825 sq. ft. and a minimum depth of 9 ft., cleared the water for a pump station of some 6,000 gal. per min. capacity. The total skimming area was 4,950 sq. ft. and the volume 44,550 cu. ft. The bottom of each settling sump was sloped on a 3.1 per cent grade to a sludge outlet that connected with a tunnel beneath leading to a sludge sump. Unlike this design, most sumps have inadequate settling capacity.

The sump is placed below the level of the pump station and in close proximity. The suction pipe extends to the pump station with take-offs

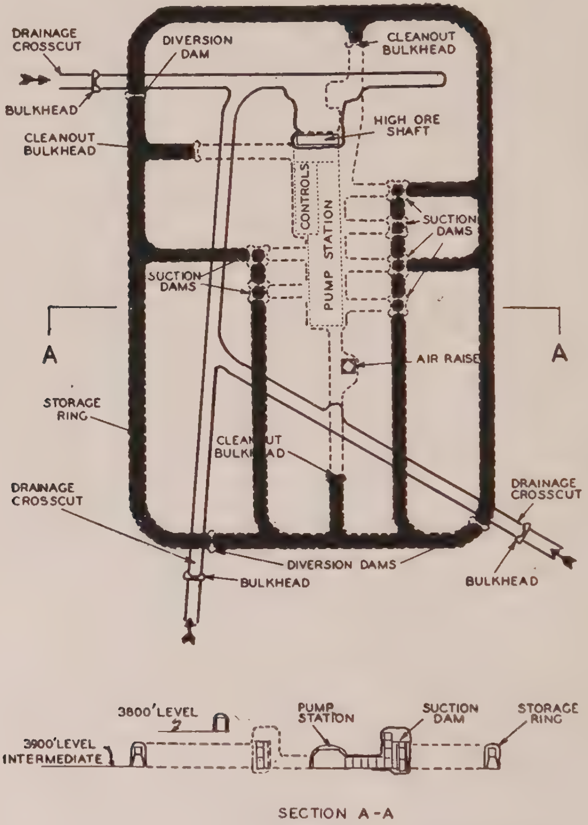


FIG. 109.—Plan of sump and pump station at High Ore shaft, Butte. (*M. McCanna, Compressed Air Mag.*)

to each pump, or a narrow ditch extends from the sump along the axis of the station, separate suctions extending into it from each pump. As suction lift is limited to about 15 ft., the depth of the sump and its connecting ditch is limited to 15 ft. below pump inlets. Two other arrangements are advantageous where heavy flows have to be handled. In one (Fig. 109) the pumps are placed below the level of the bottom of the sump and receive gravity flow through their inlets. A pipe and valve control the flow to each pump. In the other the main sump is placed below the pump station and a smaller sump is placed above to afford pressure flow to each pump. Auxiliary low-head pumps of the vertical type deliver the water from the main sump to the auxiliary sump. In still another arrangement the lower main sump delivers to a ditch so placed that vertical low-lift pumps deliver to the suction of each main pump. The low-lift pumps are of the submerged type. The gravity flow, or low-lift pump to pump, is necessary where hot water is to be handled.

Drainage must be almost continuous in wet mines. Pumping capacity is divided between two or more units, so that repairs may be made on one unit while the other is operating. Provision is also made for an auxiliary power source. As an additional precaution, sumps are designed to have sufficient capacity to allow a considerable time interval before the flooding of a pump station can take place. In some mines the pump station is placed above the lowest level, which in emergencies can be flooded. Where flooding is anticipated, development openings at pump stations are provided with pressure doors and dams with valved outlets, so that sudden water flows can be shut off and controlled. In a few cases a drainage shaft and its pump stations are sealed off from lower development workings by dams, pressure doors, or concrete bulkheads.

Sump capacity at the lowest pumping station should be large where relatively small flows are current. At least an 8-hr. flow should be provided and a 24-hr. capacity may be desirable. Large sump capacity has the advantage that pumping can be concentrated on the off-peak power loads of the mine plant. Under such conditions pumping time may be from 8 to 16 hr. and the pumping capacity arranged accordingly. Table 38 gives sump capacities for several flows. The cost of sump excavation varies according to conditions but may be 10 to 20 cents per cubic foot. Considerable cost can go into the excavation of a large sump. It is for this reason that unneeded development workings are used for sump space wherever possible.

TABLE 38.—SUMP VOLUME

Flow, g.p.m.	Weight of water, lb.	Cu. ft. per min.	Sump capacity, cu. ft.		
			8 hr.	16 hr.	24 hr.
100	834.5	13.37	6,417	12,834	19,251
500	4,172.5	66.85	32,085	64,170	96,255
1,000	8,345.0	133.70	64,170	128,340	192,510

To facilitate sludge removal, sump capacity is divided so that one part can be decanted and the sludge removed. Some sumps are designed with a bottom drawoff; the sludge is piped to the shaft, where it is discharged into a skip for removal to the surface. In other cases a diaphragm pump is installed and the sludge is pumped into mine cars. Removal by means of a scraper and loading into mine cars is another expedient. In some South African mines the sludge is pumped by high-pressure plunger pumps after dilution to about 4 per cent solids. In other cases the diluted sludge is pumped by one of the station pumps, the clearances of which have been considerably worn.

Suctions.—Where suction pipes connect with a suction header, each suction pipe is provided with a gate valve. The end of the suction pipe may be provided with a basket and inlet valve where each pump has a separate suction. Velocity of flow in suctions ranges from 5 to 10 ft. per sec. Practically all pumps have to be primed or filled with water before they can be operated. A flooded suction or a supply above the pump inlet is provided wherever possible. A vacuum pump, hand- or power-operated, is part of the pump station equipment where gravity flow is absent. Where a suction lift prevails, suction lines are designed in as straight a course as possible and all air pockets are avoided; tight connections are essential. Where a foot valve is used on the suction, a convenient expedient for pump priming is a small by-pass pipe and valve between the column pipe and the suction above the foot valve; water is drawn from the column to fill the suction pipe and the pump.

Columns or Discharge Pipes.—Discharge pipes of centrifugal pumps are connected to a horizontal main, which connects to the shaft columns or pipes. Each pump discharge is provided with a check valve and gate valve so that the unit can be cut out, the check valve being necessary to prevent any backflow through the pump. The horizontal main varies in diameter from a minimum required for one pump to a maximum required by the aggregate discharge of all the pumps upon the main. A gate valve is interposed between the header and the column pipe. Two or more shaft columns are installed and are so connected with the horizontal main as to permit the cutting out of any one for repairs. Column pipes are welded into sections connected by flanges and are supported by a cast steel tee, which is carried upon I beams set in concrete across the pump compartment of the shaft. In lifts of 1,000 ft. or more, expansion joints are sometimes placed in the pipe line, dividing it into an upper and lower section, the upper section being carried upon bearers above the expansion joint. Columns are held in place by collars bolted to the timbers at intervals. In some high-lift installations the weight of the entire column is carried by the tee at the bottom and no expansion joints are used. In stage pumping where the lower column connects directly to the pump inlet, an expansion joint of some kind is essential. In some instances hangers are placed in the shaft and are designed to take up part of the weight of the column.

Strength, corrosion resistance, weight, and cost determine the pipe material. Standard, extra-strong, and double extra-strong steel pipe are perhaps the most commonly used materials. However, wood-stave, wrought-iron, and cast-iron pipe and concrete-lined raises are sometimes used. Wood-stave pipe is especially corrosion-resisting, but redwood is limited to heads up to 300 ft. and fir up to 400 ft. Wrought iron and cast iron have good corrosion-resisting qualities. Various pipe coatings and linings are in use to increase corrosion resistance. These are pipe enamels, asphalt paints, wood, rubber, lead, and cement linings. Probably wood and lead are most generally used as pipe linings. At Rio Tinto, Spain, lead-lined cast-iron pipe and rubber-lined steel tubing are used for handling acid mine water.

Pipe sizes are approximately the inside diameters for standard steel pipe, but corresponding inside diameters are somewhat less for heavier grades of pipes. Pipe sections are joined by welding, and at intervals flanged sections join the welded sections. Victaulic couplings are also sometimes used.

Maximum quantity, gallons per minute, and allowable limits of friction head determine pipe diameter. The smaller the diameter, the greater the velocity for a given flow and the greater the friction head, expressed as loss of head per 100 ft. of pipe. As the diameter of the pipe is increased there is relatively much less loss of head. Thus a 4-in. pipe and a velocity of 5 ft. per sec. gives a 4.5-ft. loss of head, while a 12-in. pipe at the same velocity gives a 1.2-ft. loss of head per 100 ft., with about 9 times the flow. Handbook tables give pipe sizes, capacities, and friction losses for pipe, fittings, and valves. For a given flow and static head, comparisons may be made of the friction losses for different pipe sizes (see Table 39).

TABLE 39.—PIPE FLOW, GALLONS PER MINUTE—STANDARD STEEL PIPE

Pipe in.	Loss of head per 100 ft.			Pipe in.	Loss of head per 100 ft.		
	1 ft.	5 ft.	10 ft.		1 ft.	5 ft.	10 ft.
1.5	8	20	28	5	160	380	550
2	16	37	55	6	260	650	900
2.5	25	60	85	8	550	1,350	1,900
3	42	100	150	10	1,000	2,400	3,500
4	90	210	320	12	1,600	3,800	5,500

For ordinary pumping, velocities of 16 ft. per sec. are taken, but for high heads and continuous pumping, such as would be required in a very wet mine, pipe velocities of from 5 to 10 ft. per sec. are in order. An increase in velocity gives a proportional increase in delivery but at the expense of greater power to overcome greater friction. At Magma (Arizona), column pipe consists of a lower section 10 in. in diameter and an upper section 8 in. in diameter for a lift of over 3,000 ft., the respective velocities at 600 gal. per min. being 2.46 and 4 ft. per sec., and at 1,200 gal. per min., 4.92 and 8 ft. per sec. At Morococha (Peru) a 48-in. concreted raise gave a velocity of 2.66 ft. per sec. at 15,000 gal. per min. and 5.32 ft. per sec. at 30,000 gal. per min. At the Carlos Francisco Mine (Peru) a 30-in. wood-stave and concreted raise gave 8.8 ft. per sec. for a maximum pump capacity of 15,400 gal. per min.

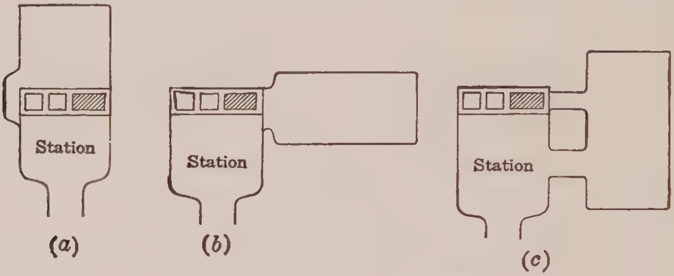
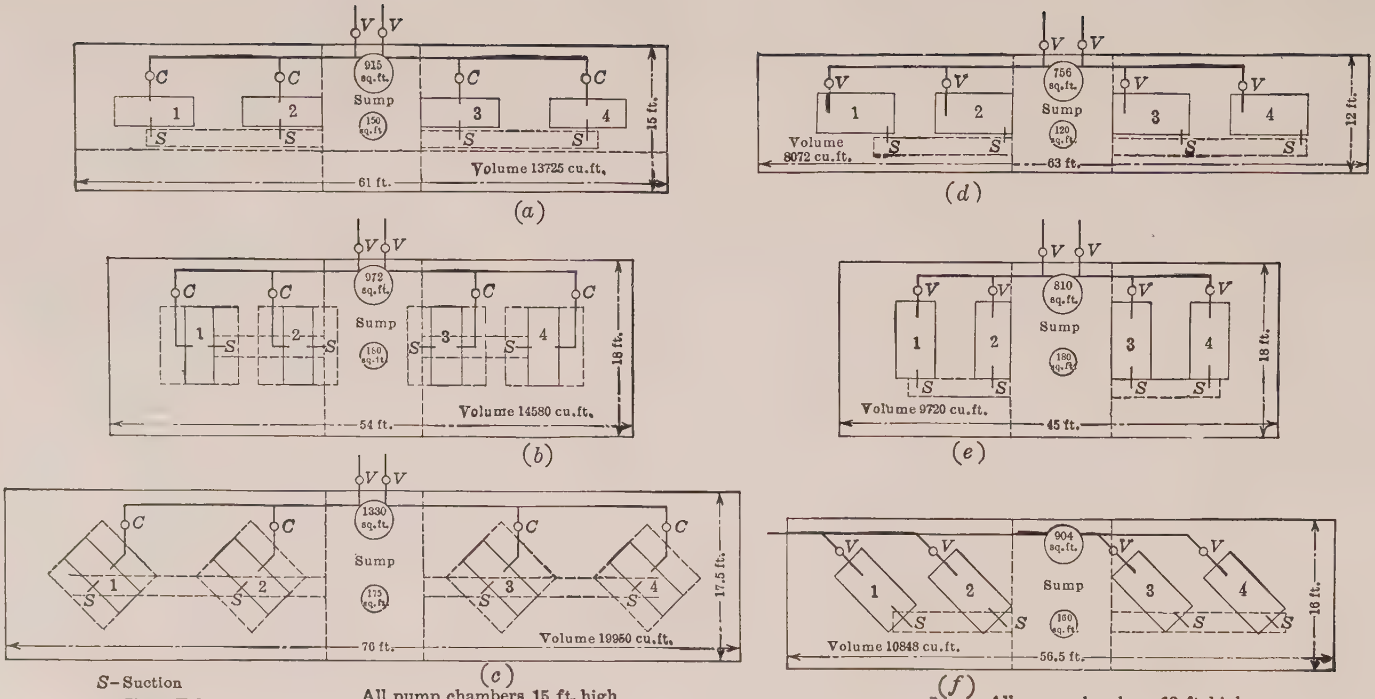


FIG. 110.—Positions of pump chamber.

The Magma pump column is an interesting example. This column is supported upon a 900-lb. cast-steel tee and extends vertically from the 3,600-ft. level to the 374-ft. level; the bottom portion consists of 830 ft. of 10-in. pipe, 80.1 lb. per ft., 0.75-in. wall; the next is 840 ft. of 8-in. pipe, 43.38 lb. per ft., 0.5-in. wall; and the top section is 1,523 ft. of 8-in. pipe, 28.55 lb. per ft. and 0.322-in. wall. All sections consist of welded pieces joined by steel flanges, the two lower sections being of 60-ft. pieces, the top one of 200-ft. pieces. Flanges are male and female with corrugated copper gaskets. The pipe was received in 30-ft. pieces and welded into the longer lengths at the surface. Total weight of the column is 118 tons. No expansion joints were used. Each pump unit has a capacity of 600 gal. per min. The pipe is held in place by steel clamps and guides at spacings of 20 to 30 ft.

Suction pipes and headers are independently supported so as to remove strains from pump casings. They are provided with drains for emptying and cleaning. By-pass pipes are provided for plunger pumps so as to reduce starting loads on motors. All piping runs are designed with the fewest possible changes of direction. Welded elbows and other fittings have displaced cast-iron fittings in headers.

Pumping Stations.—Present practice favors a narrow pump room in which centrifugal pumps can be arranged as a single or double row with short branches to an overhead header or main connecting to the column pipe. The room is placed in proximity to the pump compartment of the shaft (Fig. 110). Connections with the sump are by suction header, ditch, or tunnel. In the High Ore pumping station (Butte), crosscuts



S–Suction
C–Check Valve
V–Valve

All pump chambers 15 ft. high
4 units - each 500 gal. multiplunger pump
vertical type

All pump chambers 12 ft. high
4 units - each 1200-1500 gal.-3 stage
turbine pump

FIG. 111.—Plans of pump chambers. (Numbers refer to separate units.)

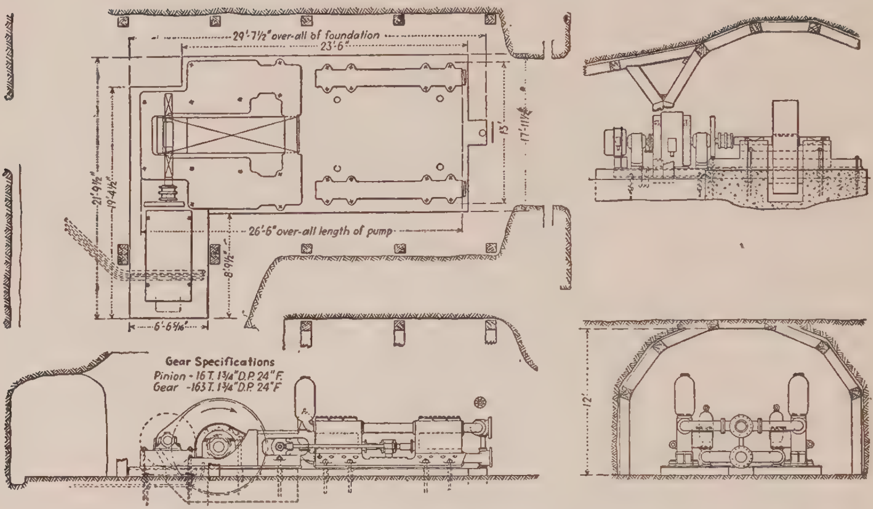

29'-7½" over-all of foundation
23'-6"
13'
17'-11¾"
21'-9½"
19'-4½"
26'-6" over-all length of pump
8'-9½"
6'-6⁵⁄₁₆"
Gear Specifications
Pinion - 16 T. 1¾" D.P. 24" F.
Gear - 163 T. 1¾" D.P. 24" F.
12'

FIG. 112.—Plan and section of part of the Magma pump station. (*E. E. Slack, Eng. Min. Jour.*)

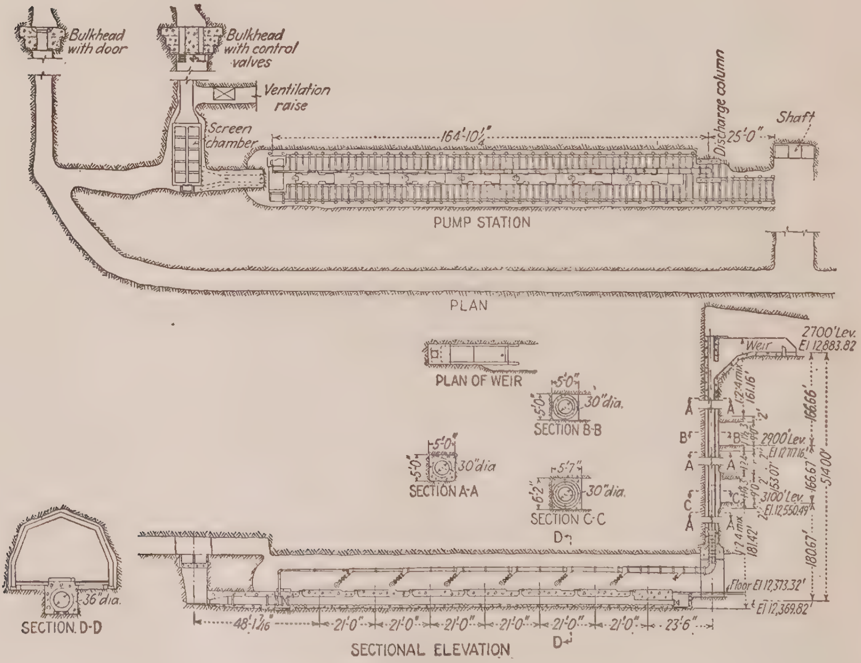


FIG. 113.—Pump station, Carlos Francisco mine. (*R. H. Misener, A.I.M.E.*)

connecting with a ring sump surrounding the station are closed by concrete dams through which the inlet pipes extend.

The width of the pump room is such as to give access to the pumps and sufficient floor area for dismantling. An overhead crawl-and-chain block or a traveling crane is placed above the pump line. The narrow width simplifies ground support where this is necessary. Concrete lining, guniting, and timbering are used according to conditions. The sealing off of drip water by guniting, tight lagging, or other means is desirable. The number of pump units and the area necessary for the electrical control panels determine the length of the station. Where tight ground strong enough to support itself over a wide span without timbering is available, the pump units are arranged transversely to the long axis of the room. In large stations the height of pump rooms is from 12 to 15 ft. in the clear. Figure 111 shows several arrangements. Figure 112 is a plan and cross section of the Magma pump station. Figures 113 and 113*a* show large stations designed for centrifugal pumps.

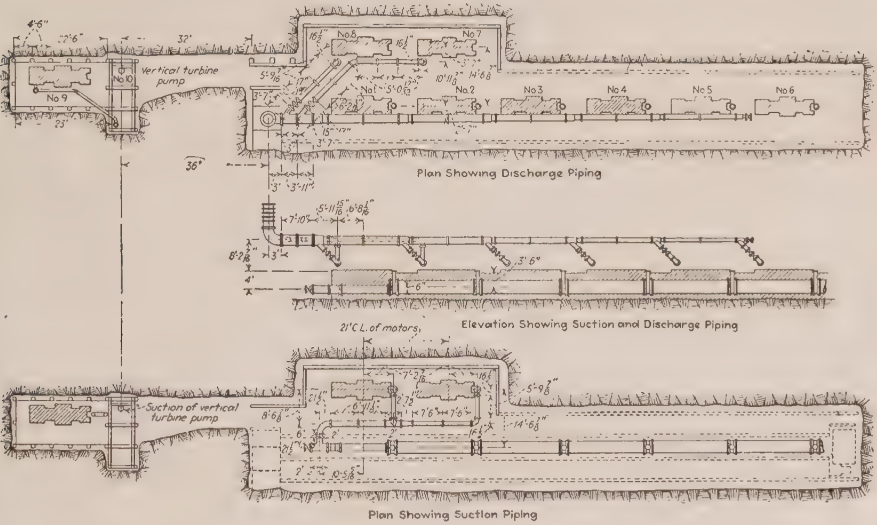

4'6"
12'6"
32'
Vertical turbine pump
No 10
No 9
23'
No 8
No 7
No 1
No 2
No 3
No 4
No 5
No 6
36"
Plan Showing Discharge Piping
Elevation Showing Suction and Discharge Piping
21' C L. of motors
Suction of vertical turbine pump
Plan Showing Suction Piping

FIG. 113*a*.—Pump station, Morococha unit. (*A. C. MacHardy, Eng. Min. Jour.*)

Provision for ventilation is necessary where the station is a large one. Motors discharge a considerable amount of heat in proportion to the power used. A kilowatt-hour is equal to 3143 B.t.u., and if 10 per cent is dissipated as heat the result is 314.3 B.t.u. Motors would thus liberate 31,430 B.t.u. for each 100 kw.-hr. The maximum power demand in a given station would determine this heat factor. Large pump stations are best ventilated by a raise connecting with the return air of the mine. Where this is impracticable, fan ventilation becomes necessary.

Some examples of pump station dimensions are given in Table 40.

TABLE 40.—PUMP STATION DIMENSIONS

Mine	Cap., g.p.m.	No. & type of pump	Floor area, sq. ft.	Dimensions, ft.			Volume, cu. ft.	Cu. ft. per 1,000 gal.
				W	L	H		
Junction Shaft (Ariz.)	4,750	4 H-D	3,827 (each)	43	89	14	32,250	11.0
Magma (Ariz.)	1,200	2 H-D	2–2,440	21	38	12	29,280	24.4
Ajo (Ariz.)	987	1 H-D	983	19	47	15	13,395	13.5
Water supply	1,325	1 W-H-D	983	19	47	15	13,395	10.1
Butte—High Ore (Mont.)	6,000	4 4-SC	2,832	28 20	118	12	33,984	5.7
Carlos Francisco (Peru)	17,500	7 4-SC	2,970	18	165	15	44,550	2.5
Morococha (Peru)	22,500	9 4-SC	4,223	20 plus offsets	170	13	54,899	2.4
Randfontein Estates (S. Afr.)	5,764	4 13-SC	3,440	43	80	15	51,600	8.9

NOTE.—H-D is horizontal motor drive duplex double-acting pump; W-H-D is Worthington horizontal duplex; 4-SC is a 4-stage centrifugal; left-hand figures refer to the number of pump units. Pumping heads are Junction 2,200; Magma 3,304; Ajo 1,176; Butte 1,100; Carlos Francisco 550; Morococha 520; and Randfontein 2,600 ft.

Pumping capacity of a given station is determined by the average quantity of water per day to be handled and by the off-peak power hours available. Off-peak power may be available for 8 to 16 hr. per day. Average quantity divided by off-peak hours and this by 60 would give the gallons per minute. The selection of the size of the pump unit and the number of units for handling the gallons per minute depend largely upon its magnitude. The sump storage would also be a factor. A thousand gallons per minute could be handled by three 500-gal. per min. units, one being kept in reserve; but with a large sump storage two 500-gal. per min. units might suffice. Maximum flow as contrasted with average flow would necessitate additional units sufficient to handle this and also to provide reserve capacity. An analysis of the water inflow over the year, taking years of exceptional rainfall into consideration, would be necessary to reach a decision. Water flow is also related to the extent of development. As the workings on a given level expand, an increased water flow as compared to that experienced when the level was started might reasonably be expected. Units of one size are installed, and in stage pumping the pump stations are standardized with reference to each other in respect to size of pumps, motors, and control equipment.

Economic details of station design include the probable life of the station, the quantity of water (minimum, average, and maximum), the head or lift, the comparative size of stations for different types of pumps, comparative costs of installation and operation, and efficiency for various types of pumps. Single-lift *vs.* stage pumping is also considered.

Power Cables.—Power cables are installed in a shaft compartment and occasionally in a diamond-drill hole or other bore extending from the surface to the pump chamber. Marine-type cables are used. They are armored and are often coated with asphalt as an additional protection against water. For severe acid conditions an armored lead-sheathed cable is installed. Rubber-covered wire in conduit may also be used and is less costly. Insulation for 2,200 volts and higher is provided. All such cables are protected against overload and are grounded according to standard specifications for such installations. Two or more circuits are usually installed. Cables are supported at intervals of 15 ft. by wide clamps and at shorter intervals where ice accumulates in a shaft. Potheads and junction boxes are protected against water entrance. Power cables terminate in switch panels, which connect with bus bars and individual motor controls of the manual or automatic compensator type.

Automatic Pump Controls.—Electrical and mechanical control apparatus has been highly developed for motor-driven centrifugal pumps, with the result that the pump station and a number of pump stations in series can be automatically operated. Barrett, Haentjens and Company of Hazelton, Pa., developed much of the apparatus following the pioneer work of J. T. Jennings, about 1920, in the anthracite coal districts. The control units and accessories are now standardized equipment. It is to be expected that electronic controls may improve mechanical arrangements now in use.

The arrangement of the automatic unit for a centrifugal pump operating under a suction lift is shown in Fig. 114. The initiatory device is a float or floats in the sump that operate electrical switches for starting or stopping the pump according to the water level in the sump. The parts of the system are a priming pump integral with the suction pipe, a priming valve placed at the highest point upon the pump casing and connected with the priming pump through a vacuum breaker, a vacuum regulator connected to the suction pipe, and a pressure regulator connected with the outlet side of the pump. The float switch starts the priming pump, which exhausts the air from the pump casing and suction pipe, the rising water filling the vacuum breaker and lifting a float (in the vacuum breaker) which closes the vacuum-breaker switch and at the same time starts the pump. During priming, air is drawn through a number of small holes in the stem of the priming valve. They offer little resistance to the passage of the air, but as soon as the pump starts, the resistance to the passage of water is increased and the priming valve closes the connection between pump casing, vacuum breaker, and priming pump. When the pump is idle a spring keeps the priming valve open. The float chamber of the vacuum breaker drains through an orifice so proportioned as to empty in about 15 sec. During this interval the pump comes up to speed and the pressure regulator closes the operating circuit of the pump motor; the emptying of the float chamber of the vacuum breaker opens

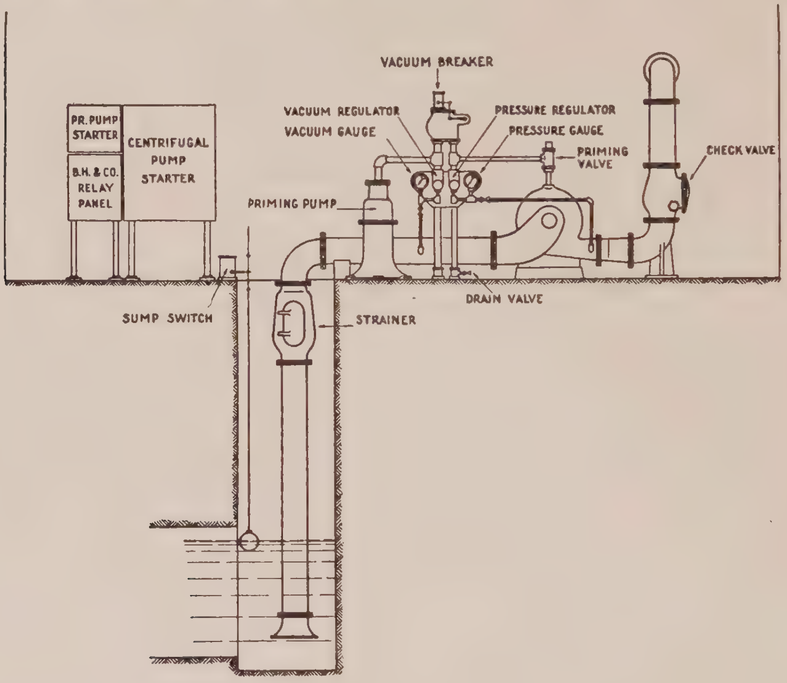


FIG. 114.—Automatic pump control. (*Barrett, Haentjens & Co.*)

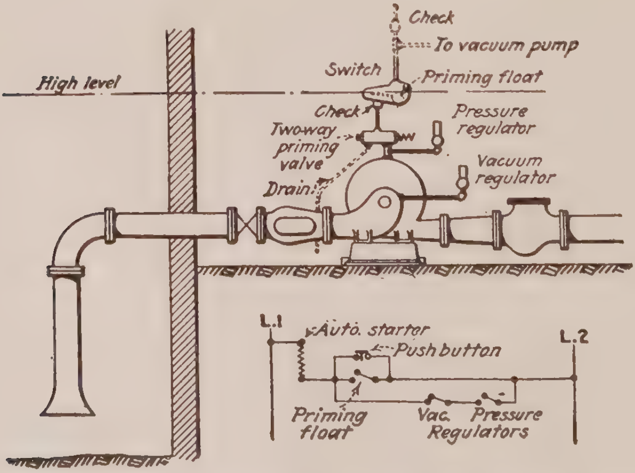


FIG. 115.—Automatic starting and stopping control for pump operating on high-level sump. (*Eng. Min. Jour.*)

its starting switch, leaving the vacuum breaker ready to repeat its function. Both vacuum and pressure regulators control the operating circuit of the pump motor: If the pump pressure falls below normal the pressure regulator switch opens and the pump motor is shut down; if the vacuum should rise too high, the vacuum regulator opens its switch and closes down the pump motor.

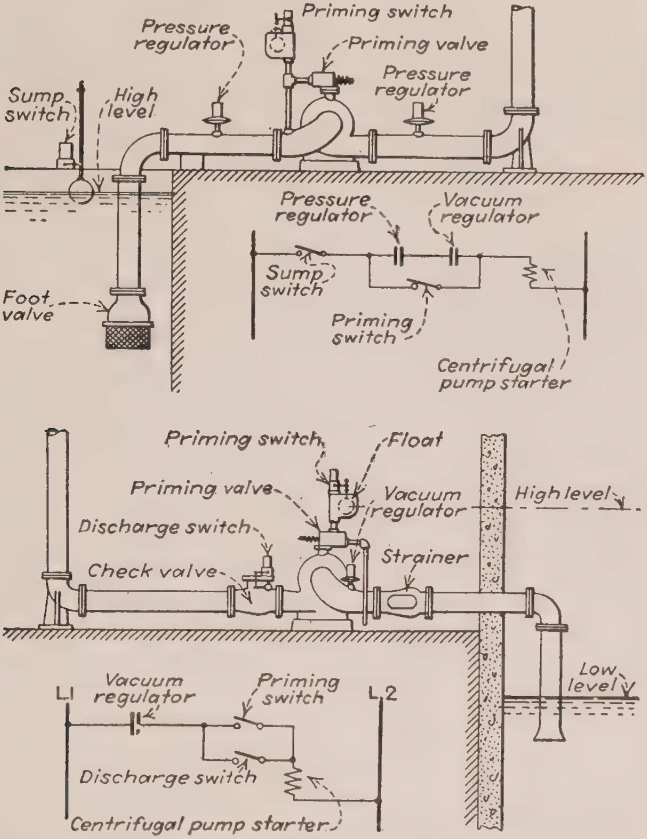


FIG. 115*a*.—Automatic pump control; priming is accomplished by a foot valve and suction lift; automatic pump control for high-level sump. (*L. Weaver, Eng. Min. Jour.*)

The control panels (Fig. 114) include the priming-pump starter, the centrifugal-pump starter, and a relay panel. Time relays are used to limit the time during which the primer can run and the number of unsuccessful attempts to start the centrifugal. Too many of these results in the locking out of the unit and closing the alarm circuit. The alarm can be of any type and located at any point. In addition, lockout relays operated by thermostats can be included to guard against overheated bearings. Means are provided also for manual operation. Gages for both vacuum and pressure regulators facilitate the adjustment of these valves. Graphic recording instruments are also useful in recording the pump service. An alternate for the pressure regulator is a switch operated by the flap valve of the check valve: When the check valve is open, the switch closes the power circuit to the pump motor; when it is closed, the power circuit is open and the motor is shut down.

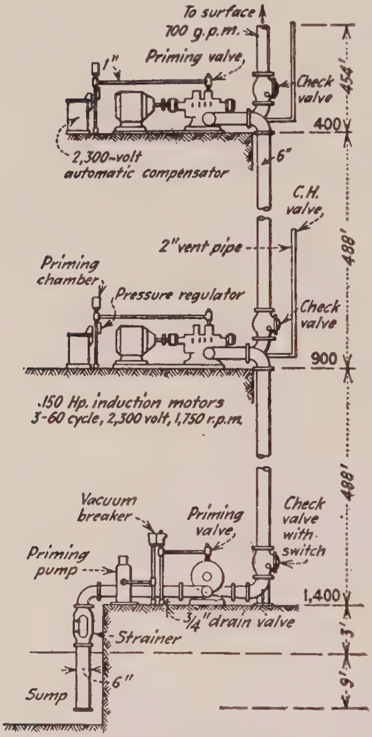


FIG. 116.—Automatically controlled pumps in series, Burra Burra shaft installation. (*L. Weaver, Eng. Min. Jour.*)

Figure 115 shows the arrangement for automatic control for pumps operated under gravity head. A priming float chamber is placed at a height to correspond with the high-water level of the sump, water rising through the intake and pump casing. A two-way priming valve lifts the priming float and closes the starting switch, bringing the pump motor up to speed. As the pressure builds up in the pump casing, the priming valve closes, the water in the float chamber draining off through the two-way valve. The pressure regulator closes the power circuit and opens the priming float switch, and now the pump is in operation. A float and switch can be used to shut down the unit when the water falls below a low level in the sump, or closing down may be accomplished by the vacuum regulator. When the pump must be operated below gravity level, a connection is provided for a vacuum pump. Switchboard panels and relays are provided with motor starter, thermostats, and alarm circuits. Figure 115*a* shows later designs.

Figure 116 shows an arrangement for the operation of three pump stations in series at the Burra Burra shaft of the Tennessee Copper Company. Two-stage balanced-unit 700-gal. per min. centrifugal pumps, driven by 150-hp. 2,300-volt motors, are connected in series from the 1,400-ft. level of the mine to the surface. The lowest pump is started by a hand compensator. Pumps on the ninth and fourth levels are provided with priming valves, priming chamber, and automatic starter; all pumps are provided with pressure regulators and flow switch with load relays for the ninth- and fourth-level compensators. The flow switch is operated by the check valve and shuts down the motor when the pump fails to deliver water. The pressure-regulator relay shuts down the motor if the discharge line breaks or the check valve sticks; the load relays function to stop the upper pumps if the lower one fails or to prevent the starting of the upper-level pumps until the lower pump has been started (L. Weaver).

COST OF MINE DRAINAGE

Drain Tunnels.—Gravity drainage is lowest in direct cost. Drain ditches are usually included in the development cost; maintenance involves cleaning at intervals and is nominal in amount. Where drain tunnels are constructed, drainage includes the cost of the tunnel, its maintenance over the useful life of the tunnel, and interest and amortization charges. Tunnels cost from $10 to $50 per foot, depending upon the cross section, length, and nature of the ground penetrated. Long tunnels involve large capital expenditures. The Sutro tunnel (Nevada), 20,489 ft. in length, is said to have cost $2,100,100. It was completed in 1879. The cost of the Carlton tunnel (Colorado) was estimated at $1,000,000 for a length over 32,000 ft. It was completed in 1941. Amortization of the cost of this tunnel over an assumed life of 10 years would amount to $100,000 per annum, and the interest charge at 5 per cent would amount to $25,000 per annum; maintenance would be low but appreciable.

Modern methods of tunnel driving have greatly increased the speed of construction and lowered the cost to such an extent that very long tunnels become financially possible. Tunnels such as the Carlton serve two distinct purposes, of which one is the dewatering of a vertical zone of considerable area, resulting in lowering development and exploration costs, and the other the drainage of the workings above the level of the tunnel during its useful life. It also reduces pumping costs in the mining ground below the tunnel level. The appraisal of the advantage of a tunnel in a given mine is comparatively simple, but to appraise the advantages of a tunnel for a mining district and to apportion its cost among the operating mines and expectant areas is not easy.

Cost of Pumping Installations.—Reciprocating pumps weigh more and cost more than do centrifugal pumps relative to capacity. Deep-well turbine pumps cost more than centrifugal pumps of similiar capacities and head. Motor capacity is from 1.4 to 1.5 times the theoretical power required for volume and head, and motor cost for motors 100 hp. or larger is approximately $8 per horsepower. The greater the head, the greater the cost of the motor and pump but not in proportion; the greater the capacity of a unit the greater its cost but not in proportion. For example, a 1,000-gal. per min. centrifugal pump and motor for a 1,000-

ft. head would have an f.o.b. cost of about $5,450; a 2,000-gal. per min. pump and motor for the same head would cost $9,650; while a 1,000-gal. per min. unit for a head of 2,000 ft. would cost $8,970. However, for pumps of 500 gal. per min. or larger and heads in excess of 500 ft., an approximate figure of 50 cents per 100-ft. head per minute-gallon can be taken in making estimates. The figures given do not include motor controls and are for cast iron, bronze construction costing about one-third more.

In the previous edition a table was given representing a summary of costs of pumping installations. These costs ranged from $25 to $131 per minute-gallon for heads ranging from 500 to 1,500 ft. and for several types of pumps, steam to motor-powered centrifugal. The period represented 1902–08. Table 41 gives the costs of three later installations.

TABLE 41.—COST OF PUMPING INSTALLATIONS

Mine	Date	G.p.m.	Head, ft.	Cost	Cost per g.p.m.	Cost, g.p.m. per 100 ft.
Commonwealth (Arizona)[a]	1915	1,500	485	$ 13,729	$9.15	$1.90
Magma (Arizona)[b]	1939	1,200	3,304	120,000	100.00	3.02
Carlos Francisco (Peru)[c]	1943	17,500	550	131,615	7.52	1.40

[a] *Min. Sci. Press*, Nov. 20, 1915, p. 786; one triplex and one centrifugal.

[b] *Eng. Min. Jour.*, October, 1939; two duplex double-acting plunger pumps, horizontal type.

[c] *T.P.* 1546, *A.I.M.E.*; seven 4-stage motor-driven centrifugals and accessories. The first cost of the pumps at the Carlos Francisco mine appears not to have been included.

TABLE 42.—SUMMARY OF PUMP-STATION COSTS AT CARLOS FRANCISCO MINE

Cost of pump station and installation[a]	$54,217.50
Discharge column	18,124.43
Installation of sump pump	1,172.27
Pumping	18,271.44
Engineering	1,248.53
Bulkheads	19,066.56
Drainage crosscuts	6,397.30
Miscellaneous	583.36
Enlarging ditch and concreting	12,533.67
Total	$131,615.06

[a] Approximate volume of station 44,550 cu. ft.; excavating costs 18 cents per cubic foot and timbering 9 cents per cubic foot, or total 27 cents per cubic foot; pump capacity 17,500 gal. per min.

Cost of Pumping.—Direct operating costs of a pump station include power, attendance, lubricants, replacements, repairs, and supervision. In addition, amortization based upon cost and probable life and an average interest charge of 2.5 per cent may be figured as a rental charge for the plant in any convenient time unit. It is seldom given in mine pumping costs. On the basis of 10 years' operating life and average interest of 2.5 per cent, the three examples given in Table 41 would approximate, respectively, $4.70, $41, and $45 per day.

In most cases, power is the principal operating cost. Next in order is attendance, with repair and replacements the least. Good design, installation, and maintenance, together with careful operation, result in low repair and replacement cost. Muddy or highly acid water gives greater maintenance and repair costs; clean water gives lowest maintenance. Stage centrifugals operating on clean water require opening up and inspection once in 6 months and on muddy water once in 3 months or oftener. Plunger pumps give less trouble than do multistage centrifugals on muddy water; sand in the water increases maintenance cost with both types of pumps but to a greater extent with the multistage. Tables 43 and 44 give comparative costs and operating, power, and maintenance costs for centrifugal and triplex pumps.

TABLE 43.—COMPARATIVE COST AND OPERATION—CENTRIFUGAL AND TRIPLEX
(420,000 Gal. per Day, 1,000 ft. Head—*L. Weaver*)

Cost	Pump room	Annual charge[a]	Power cost[b]	Maintenance cost[c]	Total, year
Centrifugal, 1,000 g.p.m., $2,100.	$180.00	$364.80	$5,856.20	$786.50	$6,987.50
2 Triplex, 500 g.p.m., $10,200...	1,433.60	1,861.38	4,822.75	613.20	7,297.30
Yearly difference................					$309.80

[a] Interest and amortization 16 per cent per annum.
[b] Efficiency centrifugal 70 per cent, triplex 85 per cent.
[c] $5 per million gallons for centrifugal and $4 for triplex.

TABLE 44.—COMPARATIVE COST AND OPERATION—CENTRIFUGAL AND TRIPLEX
(42,000 Gal. per Day, 1,000 ft. Head—*L. Weaver*)

Cost	Pump room	Annual charge[a]	Power cost[b]	Maintenance[c]	Total, year
Centrifugal, 100 g.p.m., $1,225..	$83.20	$209.31	$1,024.83	$76.65	$1,310.71
Triplex, 100 g.p.m., $1,550......	183.60	277.38	512.42	61.32	851.12
Yearly difference................					$459.59

[a] Assume 16 per cent.
[b] Efficiency of centrifugal 40 per cent; triplex 80 per cent.
[c] Per million gallons.

Size of pumping plant, number of lifts, and continuity of service determine the relation between power and attendance costs. Pumping against a lift of 2,600 ft. by 13-stage centrifugals at the Randfontein Estates mine (South Africa), maximum station capacity 6,400 gal. per min., gave a cost distribution of 70, 26.5, and 3.5 per cent, respectively, for power, attendance, miscellaneous and spare pump parts; the total average cost over 8 years was 5.05 pence or about 10 cents per thousand gallons (W. G. C. Nixon). At the Burra Burra mine (Tennessee) pumping costs under automatic control gave 59.4, 30.4, and 10.1 per cent for power, labor, and materials, respectively, with a cost of 10 cents per thousand gallons against a 1,440-ft. head; the amount pumped in 181 days was 39,820,000 at a cost of $3,946.10, of which labor was $1,200, materials $400, and power $2,346.10 (A. H. Hubbell). In the first of these two examples settlement reduced the solids in suspension to 0.0033 per cent; in the second the solids were 0.057 per cent, acidity being 120 parts per million; in the first a single lift and in the second three lifts totaling 1,440 ft. were employed. At the Burra Burra mine L. Weaver states that 220,000 gal. per day were pumped in slightly less than 5 hr. Only one man was necessary to operate the three stations, and a power efficiency of 67 per cent was obtained.

Average quantity of water, head, and over-all efficiency determine power input that can be converted into kilowatt-hours (1 kw.-hr. equals 2,655,000 ft.-lb. or 3,413 B.t.u.). Tables similar to Table 45 may be computed for estimated over-all efficiency, a specific head, and various water quantities. Given power cost per kilowatt-hour, the approximate cost for power can be figured from the table for the unit quantity 1,000 gal. per min. or 60,000 gal per hr. Taking into consideration the number of shifts per day required for pumping, the cost of labor for a given pumping layout can be estimated. With automatic pump control, attendance can be cut to one man per shift for several pumping stations

TABLE 45.—POWER IN KILOWATT-HOURS REQUIRED FOR PUMPING 60,000 GAL. PER HR.

Head, ft.	Over-all efficiency, per cent				
	40	50	60	70	80
200	94.2	75.4	62.6	53.9	47.1
500	235.7	188.6	156.5	134.8	117.8
1,000	471.5	377.2	313.1	269.7	235.7
1,500	707.2	565.8	469.6	404.5	353.6
2,000	942.9	754.4	626.1	539.3	471.4
3,000	1414.5	1131.5	939.2	809.0	707.2

TABLE 46.—ESTIMATED PUMPING COSTS PER 1,000 GAL. PER 1,000 FT.

Hr. per day	Gal. per day	Kw.-hr. per day	Cost per 1,000 gal.		
			70/30	60/40	50/50
8	480,000	2,504	$0.074	$0.086	$0.104
16	960,000	5,009	0.074	0.086	0.104
24	1,440,000	7,514	0.074	0.086	0.104

NOTE.—Over-all efficiency assumed at 60 per cent. Power cost assumed at 1 per cent kw.-hr. Ratios given in last three columns are power cost to other direct expense but not including depreciation or similar charges.

in series. Where drainage is incidental in amount, shift bosses often start and stop the pumps. Pump stations of large size equipped with plunger pumps require at least one man per shift operation.

Pumping 1,000 gal. per min. against a 1,000-ft. head requires 313 kw.-hr. power input at 60 per cent over-all efficiency. Table 46 gives quantities and estimated costs for one, two, and three shifts per day.

Pumping costs at the Ajax mine, Cripple Creek, for July, 1936, were given as $3,584.38 for 3,000 gal. per min., 400-ft. head; labor cost was $538.75, compensation insurance $35.29, supplies $90.14, power $2,524, and cost of pump assembly $396.20. The power was 282,000 kw-hr. at 0.7 cent per kw.-hr. and connected load on pumps 550 hp. at $1 or $550 per month, a total of 2,524; power cost was about 0.9 cents per kw.-hr. The cost per 1,000 gal. per min. was 2.7 cents (*Information Circular* 6982, *U.S. Bureau of Mines*). Comparatively few pumping costs are available in company reports, as drainage cost is usually represented as cost per ton of ore mined rather than in amount of water removed. Drainage cost per ton of ore produced is the important unit in the case of a given mine. Where much water must be removed relative to ore tonnage, drainage installations are designed for highest efficiencies and operation is adjusted to give the lowest unit costs. This is especially the case where power costs are also high.

SPECIAL PROBLEMS

Unwatering Mines.—Mines that have been allowed to fill, either because of temporary abandonment or stoppage of operations, present a problem somewhat different from ordinary drainage operations. Where there is an accurate map and a general knowledge of the conditions, the probable total volume of water filling the workings can be computed. If information is at hand concerning the quantity of inflow, it is possible to compute the time required for the unwatering operations. The total time in days is given by the equation

$$T = \frac{V}{V_D - v}$$

T = total time in days.
V = volume of water in mine.
V_D = daily volume removed by unwatering appliances.
v = daily inflow.

Units of volume may be either gallons or cubic feet.

The equation makes it evident that unless the volume removed by the unwatering appliances is relatively large as compared to the daily inflow, the time required for unwatering will be excessive. Hence every facility that can be brought into use is essential. The most convenient method, if the mine is equipped with hoisting facilities, is to equip the hoist with the largest water skips that can be handled satisfactorily. This is done with as little delay as possible and water hoisting is started. If an air compressor is available, air lifts are installed up to the capacity of the compressor. These lifts should be designed to unwater to the maximum depth. For example, if the shaft is clear and the depth of the mine is 1,000 ft., it would be possible to use air lifts to a depth of about 750 ft. Once work is started it should be pushed continuously, three shifts per day. As soon as a pump station is uncovered, its pumps should be overhauled and prepared for operation. The station should be put in service as speedily as possible. Figure 117 is a chart showing time required to remove 100,000,000 gal. at various pumping rates.

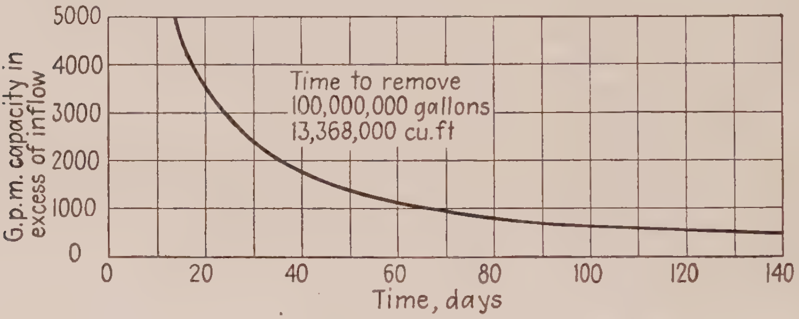


FIG. 117.—Theoretical time required to unwater 100,000,000 gal.

The deep-well turbine pump has important advantages in unwatering mines and is widely used for this purpose as well as for emergency use when large water flows are tapped. Its over-all efficiency, 60 to 70 per cent, is greater than the air lift under the most favorable circumstances. The submersible deep-well unit has still greater advantages. Deep-well turbines have large capacities and will operate under heads of 500 ft. or more; they occupy little space in a shaft compartment; the driving head is readily accessible, being at the discharge end. Priming is unnecessary.

A deep-well turbine, 3,200 gal. per min., 16-in. column in 10-ft. sections, 18 stages, 600-hp. 2,300-volt motor, operating at 1,160 r.p.m., was used in 1940 to unwater the Rosiclare mine (Illinois), which had been flooded since 1924. The lowest bowl of the pump was placed at 640 ft. below the surface discharge. Total weight of the pump was 94,000 lb., of which 83,000 lb. was suspended weight. Sixty days was required to lower the water to the 612-ft. level; the maximum quantity of water removed amounted to 4,500,000 gal. per day.

Dewatering in Advance of Development.—During the development of the Old Dominion mine (Arizona), deep-well turbines were used in boreholes to unwater levels in advance of development. At Park City (Utah) (Fig. 118), the Park City Consolidated Mines used five-stage

2,250-gal. per min. deep-well turbines in a cased 23-in. dia., 520-ft. deep borehole for dewatering in advance of development. A 10-in. spiral-welded Victaulic-coupled pipe in 10-ft. sections was installed. Subsequently a second borehole, 659 ft. in depth, was drilled and cased. Two submersible pumps in series were operated in the second bore. The combined capacity approximated 4,500 gal. per min. against a head of 500 ft. A threaded and coupled 12-in. standard pipe was used for a column, the discharge of the lower pump being received in a 16-in. dia. 75-ft. outer column, within which was submerged inlet of the upper unit. The series unit was operated 90 days and in 1940 pumped 4,300 gal. per min. against a 480-ft. head. The water table was lowered 175 ft. The discharge of the turbines was pumped from a sump on the 1,050-ft. level

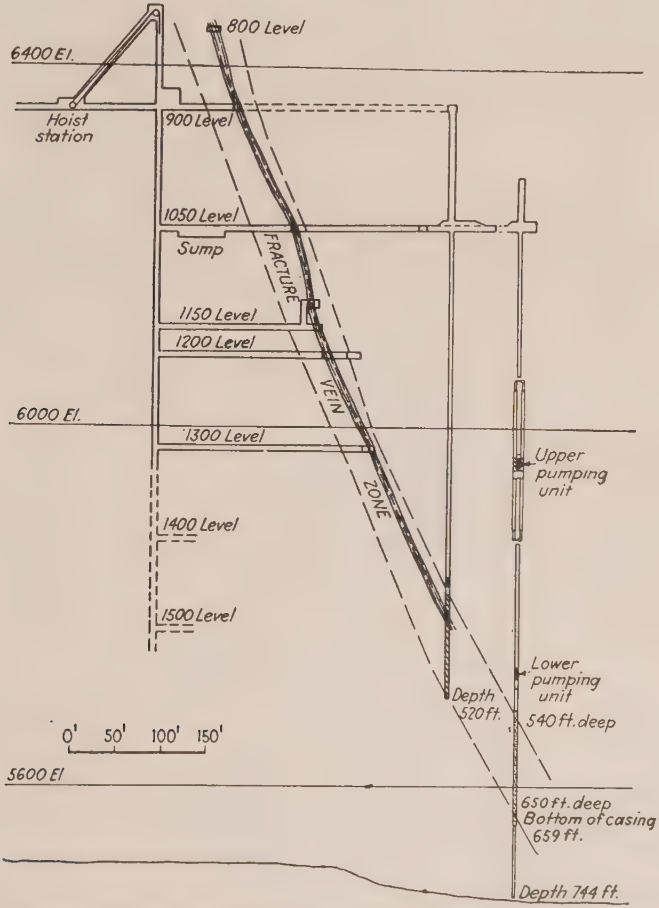


Fig. 118.—Advance dewatering at Park Utah Consolidated. (*G. M. Wiles, A.I.M.E.*)

to the 900-ft., or drainage level, by six centrifugals and one close-coupled 4,000-gal. per min. deep-well turbine pump.

Drift mines are sometimes drained by means of bores within which are installed deep-well turbines. Drainage in advance of development lessens the difficulty of shaft sinking and materially lowers the cost of development.

Tapping Neighboring Workings.—Neighboring flooded workings are often tapped by driving a working into the flooded zone. This must be carefully done, since it is dangerous work. Careful planning before connections are made is essential. Foreknowledge of the extent and position of the flooded workings is desirable. After the connection has been well started a bulkhead sufficient to withstand the hydrostatic pressure on the flooded side is constructed. The bulkhead is constructed of concrete and is provided with doors and valves and pipes for drawing off the water behind it. As the working is extended a bore is always kept in advance of the face in order to prevent suddenly breaking through. The final shell separating the workings is broken by a heavy blast after the miners have retreated through the bulkhead. Tapping by means of diamond-drill holes is safer. The diamond-drill hole is protected by a short length of pipe with plug valve attached, either cemented in the hole or secured by bolts as shown in Fig. 119.

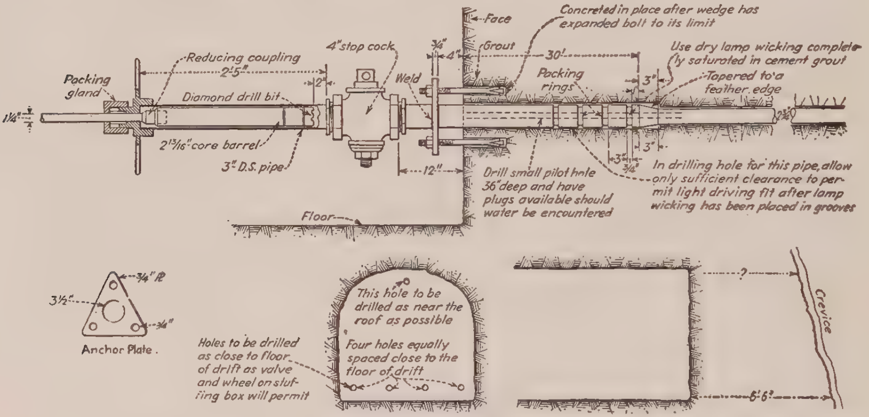
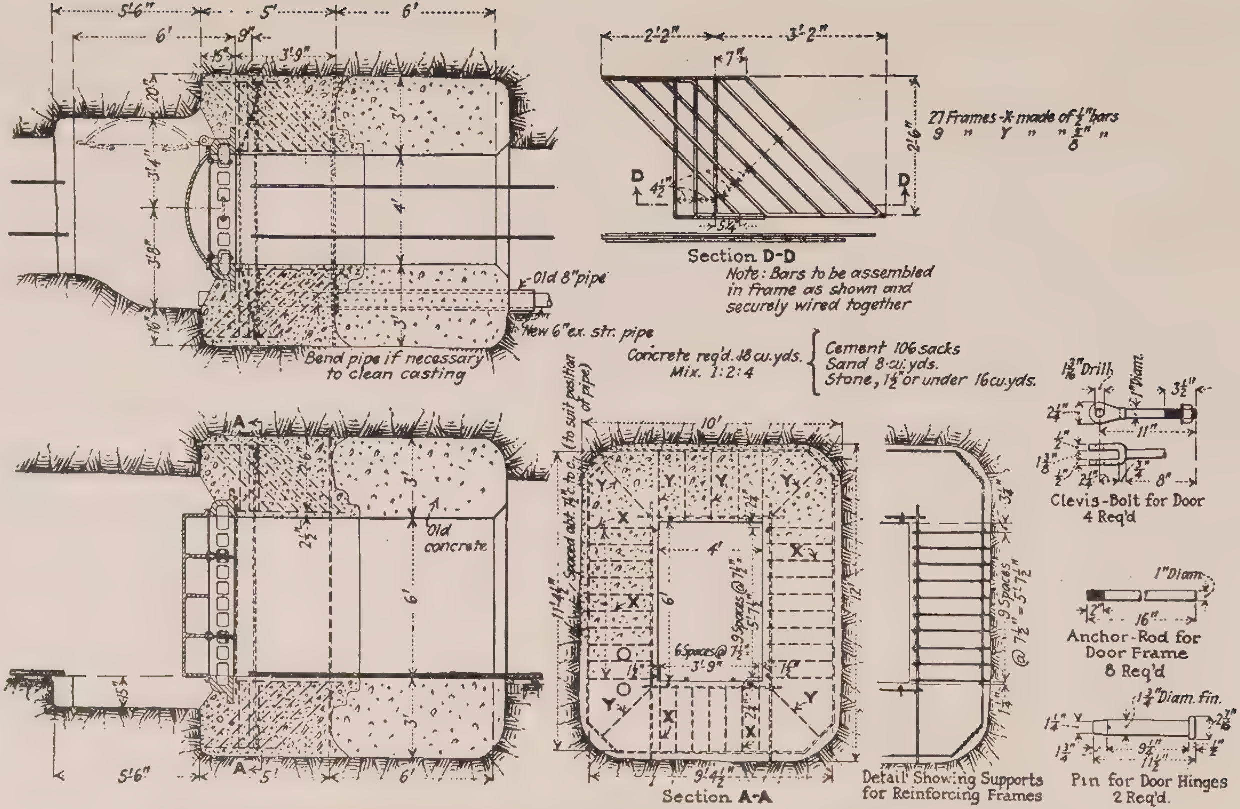

Face
Concreted in place after wedge has expanded bolt to its limit
Use dry lamp wicking completely saturated in cement grout
Reducing coupling
4" stop cock
3/4"
4"
Grout
30'
Packing gland
2'5"
2"
Weld
Packing rings
3"
Tapered to a feather edge
1 1/4"
Diamond drill bit
2 1/4"
2 13/16" core barrel
3" D.S. pipe
12"
3"
3"
3/4"
Drill small pilot hole 36" deep and have plugs available should water be encountered
In drilling hole for this pipe, allow only sufficient clearance to permit light driving fit after lamp wicking has been placed in grooves
Floor
3/4" R
3 1/2"
3/4"
Anchor Plate.
This hole to be drilled as near the roof as possible
Four holes equally spaced close to the floor of drift
Holes to be drilled as close to floor of drift as valve and wheel on stuffing box will permit
?
Crevice
6'6"

FIG. 119.—Method of sealing drill holes against pressure water. (*W. Black, Eng. Min. Jour.*)

Mine Bulkheads and Water Doors.—Waterproof bulkheads or dams are often used to seal mine workings which are no longer needed and which drain into development workings. Dams with pressure-resisting doors are constructed in development workings that are expected to penetrate water-bearing zones. Their purpose is to guard against heavy flows of water and to control such flows, protecting shaft and pump stations from submergence.



Note: Bars to be assembled in frame as shown and securely wired together

Concrete req'd. 18 cu. yds. Mix. 1:2:4 { Cement 106 sacks
Sand 8 cu. yds.
Stone, 1½" or under 16 cu. yds.

FIG. 120.—Details of a bulkhead to hold high-pressure water. (*J. S. Maffeo, Eng. Min. Jour.*)

The dam is a concrete stoping or plug (Fig. 120), which is keyed into solid ground and grouted to make it waterproof. A pipe with gate valve and a manhole are sometimes placed in the structure. The dam is of sufficient thickness to resist the maximum water pressure that might be developed. Access is limited.

Pressure doors are designed to give access for small mine cars, the size of the door and its frame being determined by the car dimensions. Flat door designs require sufficient area or spread over the concrete surrounding the opening to take care of the pressure. Where high pressures are anticipated, cast-steel frames and cast-steel doors, oval in shape and dished, with stiffening ribs, are used. Cast-iron door frames and dished steel plate doors are used with lower pressures.

The concrete dam is wedge-shaped, with the larger area against the pressure. Two or more drain pipes with gate valves are concreted in the opposite lower corners, and provision for a ventilating pipe is made in one of the upper corners; other pipes are for compressed air, drilling-water supply, and pressure gage. All piping is designed for the maximum pressure expected. A removable section of track is provided to permit the door to swing. Watertightness is obtained by packing rings in the bearing surfaces of the door frame. In many designs the concrete plug is equal in length to the maximum dimension of the development working, which is usually the height. Where pressures are very great, the concrete is reinforced. Where strong solid rock surrounds the dam the degree of keying is less than where the rock is weak. Grouting may be necessary in fissured rock. The construction cost of the simplest of these dams is $1,250 or more; for higher pressures and more complete designs, cost ranges from $4,000 to $7,500.

Handling Hot Water.—In the comparatively rare cases where hot water must be pumped more or less trouble arises at the suction end of the pump. This may be avoided by supplying the pumps under pressure head. Probably the most adverse condition due to hot water is the saturation of the ventilating current with vapor. To reduce this to a minimum, wood-stave pipe is used in place of open ditches, and all water flumes are covered. Sumps are covered or walled off from mine drifts and shaft stations. Acid water when hot is chemically more active than at normal temperatures. Water conditioning may under these circumstances become of considerable importance.

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CHAPTER IX

VENTILATION AND ILLUMINATION

Occurrence of Gases Underground.—Gases occur mechanically enclosed in most rocks. They are particularly conspicuous in coal seams and carbonaceous sedimentaries. In igneous rocks the volume of gas present ranges from 1 to 8 as compared with the volume of the rock. In sedimentaries the range is 0.7 to 5.43 volumes.[1] Carbon dioxide is the most conspicuous gas present. Carbon monoxide, methane, hydrogen, and nitrogen are also present. The mere presence of gases as a rock constituent is of itself of no practical importance to the miner, inasmuch as these gases are either not liberated or are liberated so slowly that they play no conspicuous part in polluting the atmosphere of a mine.

However, in the Cripple Creek district, in East Tintic (Utah), and in some mining districts of Nevada, an oxygen-depleted gas, consisting principally of carbon dioxide and nitrogen with small amounts of oxygen, has been encountered in raises, winzes, and dead ends where ventilation is insufficient, in such amounts as to be highly dangerous, and a number of fatalities have occurred where miners have entered such places. In a few quicksilver mines in California, methane has been encountered in sufficient quantities to necessitate careful testing of working places for the presence of this gas. In tunnel work in certain sedimentaries in California notable quantities of methane have been encountered; in other cases excessive quantities of carbon dioxide. In the Stassfurt potash mines some gases containing a high proportion of hydrogen have been reported. Recent volcanics, carbonaceous sedimentaries, and the vicinity of petroleum-bearing formations may carry excessive quantities of dangerous gases. Accumulations of gases in old workings, abandoned mines, and unventilated mine openings are suspected, and adequate ventilation should precede attempts to enter such workings.

Coal presents quite a different phase of the subject. Coal seams liberate gas in mine workings to a greater or less extent. This is true of deeply buried coal seams rather than those close to the surface or the portions of a coal seam close to an outcrop. Gas exists in coal under more or less pressure, as is shown by the observations made in English coal mines and given as follows:

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[1] Gases in Rocks. R. T. Chamberlin, *Pub. Carnegie Inst. Washington*, vol. 106.

TABLE 47[a]

Mine	Depth of hole in ft.	Pressure in lb. per sq. in.	Time pressure takes to reach maximum, in hours
Bolden	19	425	190
	23½	381	25
	28	176	60
	32	461	60
Eppleton	3½	54	1
	7½	104½	34
	24½	204	106
	25	221	51
	37	223	174
	47	235	294
Harton	16	197	135
	27½	231	126
	37¼	296	84

[a] *Coal Age*, vol. 5, p. 202.

Close to a coal face the gas is under a pressure only slightly above that of the atmosphere. Blower gas, or the gas that fills more or less open fissures, no doubt accumulates under considerable pressure, but as soon as a connection is made with the system of fissures the pressure is rapidly lowered by the escaping gas, and near the orifice the pressure is only slightly above the atmospheric pressure.

Studies have been made of the gases liberated from crushed coal. The following analyses illustrate two examples.

TABLE 48[a]

	Sample 1. Mine No. 6, Monogah	Sample 2. Mine No. 8, Monogah
Carbonic anhydride (CO_2)	2.90	1.56
Methane (CH_4)	39.65	40.99
Oxygen (O)	10.07	9.86
Nitrogen (N)	38.11	37.62
Nitrogen excess	9.27	10.27

[a] *Bull.* 72, *U.S. Bur. Mines*, p. 37.

The analyses are not necessarily representative, but they indicate in a general way the nature of the gases contained in bituminous coal and which may be expected to be liberated in the workings of a coal mine. The principal gas liberated from coal is methane (firedamp, marsh gas). Undoubtedly incidental quantities of carbonic anhydride and nitrogen are always present.

The quantity of gas liberated within a coal mine depends upon the nature of the coal, the amount of gas within the coal itself, the number and extent of fissures and natural openings charged with gas, the areal extent of coal face exposed, the extent of old workings, and the rate of mining the coal. Necessarily these factors will vary from mine to mine, and all cases occur ranging from the coal mine in which dangerous gases are conspicuously absent or in such small amounts as to be negligible up to the mine where the quantity of gas is such as to make its operation extremely hazardous. In terms of cubic feet per ton of coal mined, the following figures illustrate the quantity factor.

Table 49

Locality	Cu. ft. gas per ton mined	Relative volume gas to coal
Germany	16–1,060	½–30
Austria	7,469	300
Wilkes-Barre, Pa	1,500	60
Anzin, France	1,377	55

In terms of cubic feet per 1,000 sq. ft. of coal face exposed the quantity factor is further illustrated.

Table 50.—Gas in Lance Collieries[a]

Northern Penn. anthracite field		Volume of methane per min. per 1000 sq. ft. coal exp.	Volume per ton mined, approximate
Red Ash		0.73	2,400
Ross		1.00	15,600
Five-foot	east side	1.67	6,400
	west side	0.23	60
Hillman	east side	1.00	3,850
	west side	2.30	2,070

[a] *Bull.* 72, *U.S. Bur. Mines*, p. 120.

The following table illustrates the quantity of methane liberated per unit of time in a part of a mine in which a known quantity of air was circulated. The quantity of air was varied from time to time.

Table 51[a]

Cu. ft. of air per min. in circulation	Methane		
	Per cent	Cu. ft. per min.	Cu. ft. per hr.
17,000	0.37	62.9	3,774
18,000	0.34	61.2	3,672
20,000	0.29	58.0	3,480
22,000	0.19	41.8	2,508
24,000	0.16	38.4	2,304

[a] Fairmount Coal Co., *Eng. Min. Jour.*, July 3, 1909, p. 14.

It should be noted that methane is liberated continuously from coal faces in a greater or less amount. Probably this amount is greatest for freshly exposed coal and a minimum for long exposed coal faces. Ordinarily the gas can be safely taken care of by the normal ventilating currents. Blowers and local liberation of large quantities of methane may render parts of a mine very dangerous for the time, and it is the practice to discontinue working in those parts until the surplus gas has been removed and the influx reduced to normal quantities. The practice of frequently determining the quantity of methane in the return air of a gaseous mine is to be recommended.

The principal danger of methane consists in the fact that it is a combustible gas and when mixed with air in certain proportions forms explosive mixtures. Such mixtures, unless removed or reduced by dilution with air, constitute a great danger, for they require only ignition for an explosion to ensue.

Gases Produced by Exhalations and Combustion.—Exhalations from men and horses introduce relatively small quantities of carbonic anhydride into the air. The oxygen content is materially lowered where the air circulation is bad. Candles and lamps also introduce carbonic anhydride, and where oil lamps are used more or less smoke pollutes the air. The dilution of these gases is not a difficult problem.

TABLE 52.—DIESEL LOCOMOTIVES IN TUNNEL OPERATIONS
(*Trans. A.I.M.E.*, vol. 153, p. 165; condensed)

Tunnel	C	C	B	B
Locomotive number	3	6	3	4.5
Engine load	Idle	Load	Load	Load
Carbon monoxide, per cent:				
In manifold	0.15	0.15	0.10	0.10
In exhaust	0.035	0.03	0.03	0.03
In tunnel air, maximum	0.002	0.009	0.0025	0.004
In tunnel air, minimum	0.0001	0.0001	0.0010	0.0006
Air flow, cu. ft. per min	25,000	18,500	25,000	45,000

NOTE.—Air-fuel ratio 20 to 1; exhaust gases cooled and scrubbed; exhaust diluted 10 times before discharge; rated horsepower 160.

The exhaust from internal-combustion engines is charged with carbon dioxide and monoxide and is a definite hazard. Most mine regulations prohibit the use of such engines underground. However, under exacting control diesel locomotives equipped with coolers, scrubbers, and diluters on their exhausts are used to an increasing extent in tunnel construction; the same may be said for autotrucks. In all such cases power ventilation is necessary, and the quantity of air flow must be sufficient to dilute exhaust gases so that their monoxide content is far below the danger limit. Table 52 gives testing results under operating conditions for diesel locomotives used in tunnel construction. In most metal-mining operations the internal-combustion engine is too hazardous for use underground.

Gases Produced by Blasting.—Apart from carbonic anhydride, which is almost universally present in the gaseous products of an explosive, carbon monoxide, hydrogen sulphide, and nitrogen oxides are sometimes present and their poisonous nature renders them particularly objectionable. Thorough detonation is essential in the use of all dynamites. Black powder always produces more or less carbon monoxide. The careful selection of explosives and their proper use reduces the amount of deleterious gases to a minimum. Dilution with sufficient air is the best method of overcoming powder fumes and of rendering them comparatively harmless.

Gases Produced by Abnormal Conditions.—Fires and explosions produce for a time an irrespirable atmosphere. Apart from the carbonic anhydride, carbon monoxide in greater or less amount is always present. Sulphur dioxide is present where oxidation of sulphides takes place. After a fire or explosion, oxygen breathing apparatus is a necessity in order to safely examine a mine or to effect rescue.

Dust Particles in Suspension.—Rock dust is in suspension to a greater or less extent in the air in a dry mine and is objectionable for hygienic reasons. Drilling, shoveling, ore handling, and blasting produce dust, and its elimination is essential. The use of sprays in drilling, wetting down ore which must be shoveled, blasting at the end of a shift, and adequate ventilation go far toward reducing the dust menace.

In coal mines, coal dust mixed with air and in the form of a suspension usually forms an explosive mixture when ignited. Flame propagation accompanies the explosion. Some coal dusts are more dangerous than others, but all coal dusts in suspension in air are dangerous. This also applies to carbonaceous dust in general. The presence of methane increases the susceptibility of coal dust and air mixtures to explode. Bureau of Mines experiments indicated the explosiveness of a mixture of as low a density as 0.032 oz. per cu. ft.; a density of 0.111 oz. per cu. ft. gives approximate maximum explosive effect. In metal mines, sulphide dust caused by blasting may produce more or less hydrogen sulphide, sulphur dioxide, and carbon monoxide in dangerous amounts in addition to the gaseous products of the explosive.

Rock dust in suspension varies in particle size and composition. The most dangerous dusts are silica, sericite, and asbestos; but all fine dusts, including coal dust, are considered to be health hazards when inhaled, and the smaller sizes, 10 microns and less, are more dangerous than the larger sizes. Dust concentrations may be measured in milligrams per cubic meter, but later practice is to determine the number of particles per cubic centimeter and per cubic foot.

Dust concentrations vary from point to point and from time to time, a number of determinations being necessary to obtain an average. One hundred particles per cubic centimeter gives a density of 2,832,000 particles per cubic foot. Permissible standards vary from 10 million particles to 15 million per cubic foot where the number of free silica particles is less than 35 per cent and 5 million particles where all dust particles are free silica. A. S. Richardson gives the average daily dust concentrations for the Butte mines, in millions of particles per cubic foot, as follows: General mine air 2.1; stopes 1.8; working faces on levels 1.4; raises 4.2; shaft stations 1.8; intake shafts and skip-loading stations 3.1, and intake shafts 0.7. Forty-one South African gold mines in 1940 gave averages in particles per cubic centimeter as follows: Main traveling ways 68; drives 122; raises 135; winzes 115; ore bins 122; machine stopes 120; hammer stopes 75; back stopes 112; shaft sinking 131; night shifts 98, and over-all average 115. Particle sizes range from 0.2 micron, the smallest visible, up to 10 microns.

Nature of Mine Air.—Analyses of the return air from metal and coal mines are infrequent and no generalization is possible. Some specific examples can be quoted. The following figures apply to a South African mine.

TABLE 53[a]

		Carbon dioxide	Carbon monoxide
a	Natural ventilation	0.552	0.011
	Power ventilation	0.149	0.006
b	Natural ventilation (av. results)	0.499	0.012
	Power ventilation (av. results)	0.127	0.005

[a] *Jour. Chem., Met. Min. Soc. South Africa*, vol. 11, p. 967.

TABLE 54[a]

Return	Place of sampling	Air current		Per cent. of gases in return air			Volume of methane per min.
		Cu. ft. per min.	Velocity, ft. per min.	Methane	Carbon dioxide	Oxygen	
Red Ash	75 ft. east fan	52,400	624	0.41	0.18	19.65	215
Red Ash	East No. 1 fan	63,300	713	0.49	0.24	19.74	3,10
Red Ash	Right slope, 150 ft. from No. 2 air shaft	159,000	914	0.73	0.18	20.48	1,161
Ross	East and west returns at bottom No. 2 air shaft.	44,200	502	1.59	0.03	20.31	703
Ross	100 ft. south of No. 2 air shaft.	57,800	359	1.14	0.07	20.28	659

[a] *Bull. 72, U.S. Bur. Mines*, p. 168.

Darton gives many examples of the methane content in the return air of coal mines. From the many tables one has been selected that represented conditions at the Nottingham mine in the northern Pennsylvania anthracite region. See Table 54.

Properties of Gases Found in Mines.—The physical properties of gases are given in Table 55.

TABLE 55

	Chemical symbol	Specific gravity air = 1	Wt. cu. ft. in lb.; temp. 32°F. and bar. 30 in.	Wt. 1 cu. meter in kilos, 0°C.-760 mm. standard conditions
Air	O + N	1.0	0.08097	1.2936
Hydrogen	H	0.069	0.00559	0.0896
Oxygen	O	1.106	0.08955	1.43
Nitrogen	N	0.968	0.07379	1.2553
Marsh gas	CH_4	0.559	0.0453	0.7218
Carbon dioxide	CO_2	1.529	0.1238	1.9714
Carbon monoxide	CO	0.967	0.0783	1.252
Hydrogen sulphide	H_2S	1.191	0.0964	1.5407
Ethane	C_2H_6	1.048	....	1.3505
Sulphurous acid	H_2SO_3	2.21	....	2.859

Properties of Gases.—Air has the following volume composition, the constituents being given in percentages:

Oxygen	20.93
Nitrogen	78.10
Argon	0.94
Carbon dioxide	0.03
Water vapor	Variable

It is tasteless and odorless, and a supporter of combustion.

Hydrogen is an exceedingly rare constituent of mine gases. It is nonpoisonous, combustible, and a nonsupporter of combustion. With air it forms an explosive mixture. It is colorless, odorless, and tasteless, and in small quantities has no particular physiological effects.

Nitrogen is colorless, odorless, tasteless, and nonpoisonous. It will not support combustion, nor can life be sustained in an atmosphere of nitrogen. It tends to accumulate in the upper portions of workings.

Methane (pit gas, or firedamp when mixed with air) is colorless, odorless, tasteless, and nonpoisonous. It will not sustain life or support combustion. It is combustible and forms an explosive mixture when mixed with air. It is common in many coal mines but is rarely met with in metal mines. A mixture containing air, coal dust, and 2.5 per cent methane will explode violently on ignition. The percentage of different gases that will form explosive mixtures with air is given in Table 56.

TABLE 56[a]

Gas	Per cent by volume	
	Upper limit	Lower limit
Hydrogen	66.4	9.45
Illuminating gas	19.1	7.9
Methane	12.8	6.1
Alcohol vapor	13.65	3.95
Benzene vapor	4.9	2.4

[a] *Comp. Air. Mag.*, vol. 20, p. 7472, January, 1915. See *T.P.* 119, *U.S. Bur. Mines.*

Carbon dioxide (chokedamp, blackdamp, heavy air—all mixtures of carbon dioxide and air) is colorless and nonpoisonous. It has a faint acid taste and smell. It will not support combustion or sustain life. The physiological effects are similar to those sustained in an atmosphere impoverished of its oxygen, namely, increased depth of respiration, gasping, oppression, panting, and finally unconsciousness (Wabner). There is a question as to the toxic properties of carbon dioxide.

Carbon monoxide (white damp, stonedamp, sweet damp—mixtures of air and carbon monoxide) is colorless and tasteless and has a faint sickly smell. It is extremely poisonous and its physiological effects are giddiness, swelling of the veins in the forehead, weakening of the sight, palpitation of the heart, unconsciousness, and death (Wabner). Mixed with air it forms an explosive mixture (1 vol. to 1.6 vol. of air to 1 to 6.7 constitute the explosive mixtures).

Hydrogen sulphide (stinkdamp) has a distinctive smell. It is very poisonous and is fortunately rarely found in mines.

Sulphur dioxide is occasionally met with as a product of mine fires in mines where massive sulphide ore bodies are found. It is irrespirable and poisonous, irritating to the eyes and respiratory passages.

Nitrogen oxides as the product of the imperfect detonation of dynamites are not uncommon. They are poisonous.

Hygienic Requirements of Mine Air.—Practical conditions are so varied that it is impossible to impose any rigid standard of purity. Mine air that does not contain in excess of 10 to 20 parts of carbon dioxide per 1,000, practically no carbon monoxide, hydrogen sulphide, or nitric oxides, and a dust content not in excess of 10 mg. per cu. m. may be tentatively considered of sufficient purity for metal mines. For coal mines the following standard is suggested; 10 to 20 parts of carbon dioxide per 10,000 parts of air, no carbon monoxide, hydrogen sulphide, or nitric oxides, a dust content of not in excess of 500 mg. per cu. m., and a methane content of not in excess of 1 part in 1,000.

Air temperature and relative humidity affect the efficiency of the underground worker; high temperature and humidity give less work. Relative humidity at any given air temperature is measured by the difference between that temperature (dry-bulb temperature, t) and the temperature of the wet-bulb thermometer (wet-bulb temperature, t_1), or $t - t_1$. When the difference is zero, the relative humidity is 100 per cent, or the atmosphere is saturated with moisture. The greater the spread between t and t_1, the smaller is the relative humidity for any given temperature; for an increase in barometric pressure the relative humidity percentages decrease slightly at high temperatures, 80 to 100°F., and more as the temperatures become lower.

Body temperature is maintained at its normal point by evaporation of the moisture exuded by the sweat glands; the drying or evaporative power of an air current depends upon its temperature, relative humidity, and velocity. At 98°F., 100 per cent relative humidity, and zero velocity, little or no evaporation takes place and the body temperature rises; when the relative humidity is lowered and air movement is brought about, evaporation begins and body temperature is prevented from rising much above its normal point. At high temperatures, low humidities, and high velocities, evaporation becomes so rapid that wind burn takes place. In hot mines wet-bulb temperatures are kept below 85°F. and air velocity is kept to not greater than 500 ft. per min. where temperatures exceed 110°F. In a general résumé of conditions in the Kolar gold mines, A. Caplan states that most cases of collapse occurred in workings below 6,000 ft.; the dry-bulb temperature was commonly in the neighborhood of 110°F. and the wet-bulb 93°F.; he further states that the wet-bulb temperature may be taken as a reliable index of the degree of comfort existing in the working places and that with a dry-bulb from 110 to 120°F., a wet bulb of 90°F. or under denotes good ventilation and comfortable working conditions, 91 to 93°F. borderline conditions, 94 to 95°F. poor ventilation, and 96°F. or over conditions highly conducive to collapse. In many hot mines humidity and temperature control are effected by air conditioning or precooling of air at some stage in its passage through the mine before it enters the hotter portions of the workings. In all such mines plenty of drinking water and salt tablets are provided.

In most coal mines the temperatures are not greatly elevated and the air while approaching saturation is sufficiently low in temperature to ensure comfortable working. In winter, on account of the air entering at low temperatures and the mine's being at a much higher temperature, there is a general drying out of the mine and in time dangerous conditions may be set up. Turning steam into the intake air and the use of water sprays are resorted to for the purpose of saturating the mine air with moisture and preventing the above condition. High humidity, even at the expense of decreased efficiency on the part of the workers, is essential in dry and dusty coal mines.

The air flow in a mine depends upon its peculiar conditions and the interpretation of such conditions by its engineering staff. It is related to the number of men underground, tons mined, dust produced, temperatures in working places, and general operating conditions underground. Mining regulations often specify minimum requirements, especially for coal mines. In Pennsylvania these are 150 cu. ft. per min. in nongaseous and 200 cu. ft. per min. in gaseous bituminous coal mines and 200 cu. ft. per min. in anthracite mines per underground worker and 500 cu. ft. per min. for each horse or mule. In South African gold mines, a minimum of 30 cu. ft. per min. per underground worker is specified by law, but the average of some 38 mines was 70.6 cu. ft. per min. per man, 70.4 cu. ft. per min. per ton mined, and 2,101 cu. ft. per min. per drill used (E. H. A. Joseph). In mechanically ventilated Lake Superior iron mines, E. C. J. Urban gives a range of 250 to 800 cu. ft. per min. per man underground. C. S. Gibson gives for Canadian gold mines an average of 284 cu. ft. per min. and a range from 150 to over 800 cu. ft. per min. G. E. McElroy gives a low of 50 to 100 cu. ft. per min. up to 200 to 300 cu. ft. per min. and for high temperatures up to 1,000 cu. ft. per min. At the Magma mine some 780 cu. ft. per min. is required to cope with high temperatures.

Tests of Mine Air.—Only those tests that can be conveniently made will be described. This subject has been very thoroughly discussed in *Bulletin* 42 of the U.S. Bureau of Mines, and to this the reader is referred for further details.

Quantity of Air.—The determination is made by the anemometer. Either the anemometer is moved across the section of the airway or a number of separate readings are made in different parts of the section and the average of the readings is taken. Area of working and corrected average anemometer velocity in feet per minute give cubic volume.

The anemometer will satisfactorily measure velocities from 150 to 2,000 ft. per min., and even velocities as low as 50 ft. per min. can sometimes be measured. The timing of the passage of a smoke cloud over a measured distance in an airway will give a moderately accurate base for calculating very low velocities. A smoke tube devised by the Bureau of Mines consists of a pressure bulb, valve, and glass tube containing granules of pumice wet with tin or titanium tetrachloride. Air is forced by the bulb through the tube and a dense white smoke cloud is produced. A stop watch is necessary for timing. Sensitive manometers and torsion vanes have been devised for measuring very low air velocities, but these are seldom used underground.

For high velocities the manometer and Pitot tube give satisfactory results and are used in ventilation surveys. The manometer may also be used for the measurement of lower velocities. The aneroid barometer is an important accessory in determining static pressures. Altimeters or precise aneroids are used in ventilation surveys to determine loss of pressure from inlet to outlet of each airway or split. Changes in static pressure, where corrected for differences in level, indicate pressure loss between the points compared. In some coal mines altimeter surveys of airways and splits are made at intervals in conjunction with regular ventilation surveys.

Temperature and Relative Humidity are determined from readings of the dry- and wet-bulb thermometers mounted in a sling psychrometer; the cloth on the wet-bulb thermometer is dipped in distilled water and the pair of thermometers are whirled until the readings are constant. A barometric reading is taken, and from the three observations psychrometric tables (see *Weather Bureau* 235, U.S. Department of Agriculture) will give relative humidity and dew-point temperature. Figure 121 shows the relation between temperature and relative humidity.

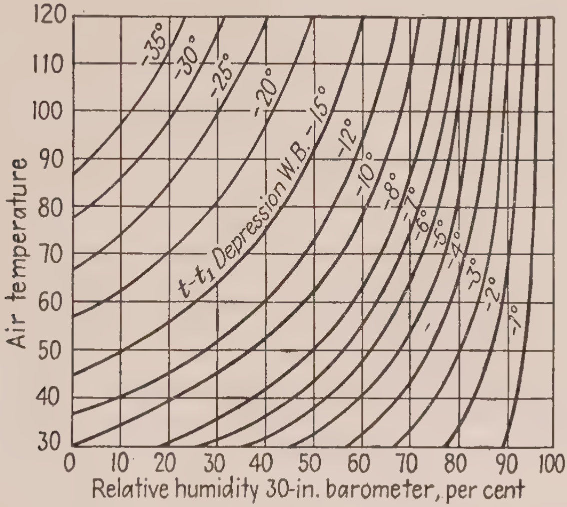

FIG. 121.—Depression of wet-bulb thermometer and per cent relative humidity at 30-in. barometer.

Air above the body temperature of 98.6°F., if unsaturated, will absorb moisture and exercise a cooling effect upon the body. The velocity of the air is also a factor in the rate of cooling. While wet- and dry-bulb temperature observations and velocity determinations give a very good indication of cooling power of mine air, some actual measure of cooling power seemed desirable, and Dr. L. E. Hill invented the katathermometer for this purpose. This instrument was used to a considerable extent but requires perhaps more skill and care to use and intrepret its recordings than are available ordinarily. As a consequence it is not widely used, and the findings of the psychrometer and anemometer are considered sufficient for the control of the ventilation in hot areas of a mine.

Dust.—Two important instruments are in use for the determination of the dust content of mine air, the konimeter, invented by Sir R. N. Kotze about 1918, and the impinger, invented by L. Greenburg and G. W. Smith in 1922. The konimeter collects the dust from a small volume of air by impingement upon an adhesive film over a glass plate; in its later development a glass disk is used and samples are taken upon successive portions of the disk. Various adhesives—mineral oil, petrolatum, and glycerin jelly—are used. After sampling, the glass disk is sometimes heated to a dull cherry red to remove film and organic matter. Then it is cooled, washed with a few drops of 50 per cent hydrochloric acid followed by distilled water and alcohol, and dried.

Dust particles are counted by a microscope at 150 diameters and dark field illumination. In other instances the konimeter samples are counted without any treatment and with normal illumination. A special microscope with an adjustable revolving frame to hold the konimeter disk so that the dust spots can be passed under the objective and easily centered is more convenient than the microscope with the standard mechanical stage. An eyepiece micrometer of special design with 18-deg. sectors dividing the field gives a counting area of 0.1 of the field of view; the number of particles within the sector limits is counted. The volume of air taken is 2.5 cc., 5 cc., or larger, in multiples of 5 cc.; the number of particles counted within the limits of a pair of sectors (2 × 18 deg.) multiplied by 10 and divided by the number of cubic centimeters taken (the volume) gives the number of particles per cubic centimeter; multiplying by 28,320 gives the number of particles per cubic foot. The microprojector is a better method of estimation. In this apparatus the magnified dust field is projected upon a translucent screen, which is hooded to cut off extraneous light. The particles are counted upon a grid containing sectors and scale at the magnification; magnifications of 500 and 1,000 are used.

The impinger (Fig. 122) is a stoppered vessel containing a known amount of alcohol or a mixture of alcohol and water. It is provided with an inlet tube, an orifice of 2.3 mm. diameter or less that discharges within 5 mm. of the bottom of the vessel, and an outlet tube with baffle. The outlet tube connects with a pump, by means of which a known volume of air to be sampled is drawn at a controlled rate through the vessel, discharging its dust content into the alcohol. The solution within the vessel is diluted to a given volume or mark and shaken, and a small portion of it is taken by means of a graduated pipette. This portion is measured into a counting cell 1 mm. in depth, the cover slip is placed, and, after an interval for settling, the counting cell is placed upon the microscope stage and the particles are counted. The microprojector with its translucent screen and a projected magnification of 1,000 diameters is also used for counting.

The thermal precipitator is still another device for obtaining dust samples from mine air. In this device a measured volume of air is drawn past a wire (heated electrically) at a controlled rate by means of a water aspirator. The dust is precipitated upon glass cover slips on either side of the heated wire. The cover slips are removed and the number of dust particles is determined by counting, either by the direct method or by microprojection. The cover slips are usually treated by heating and acid before they are mounted on a slide. The apparatus with its accessories weighs about 20 lb. A storage battery supplies the heating current of 1.2 amperes. It is said that the thermal precipitator is 100 per cent efficient for particles up to 20 microns in size.

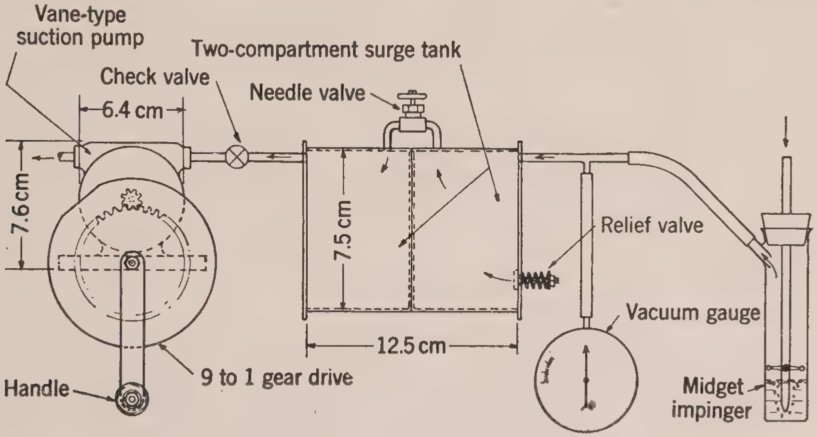

FIG. 122.—Midget-impinger sampling apparatus. (*U.S. Bur. Mines.*)

Carbon Anhydride can be determined by aspirating a known volume of the mine air through a washing bottle containing a known volume of standard solution of barium hydroxide and titrating for the excess of hydroxide. Dangerous accumulations of this gas can also be detected by the safety lamp, the flame of which shortens and gives decreased illumination until it is extinguished at about 16.25 per cent oxygen; a miner's candle will give a similar reaction, the flame becoming smaller, flickering and "looking sick"; tests are made near the floor level and near the roof. It is to be noted that this flame test indicates only oxygen depletion that may be due to the presence of carbon dioxide or to more than normal amounts of nitrogen. If combustible gases are suspected the safety lamp should be used.

Methane is approximately determined by means of the approved (Bureau of Mines) safety lamp in atmospheres where the oxygen content is above 19 per cent, as below 16.25 per cent the flame of the lamp is extinguished. In using the safety lamp, the flame is first adjusted to standard height in a methane-free atmosphere. When the lamp is introduced into the suspected atmosphere, the presence and to some extent the proportion of methane are indicated by the appearance of a blue gas cap and its height above the base of the lamp flame. At 1 per cent the round-wick Kohler lamp gives a gas cap 0.3 in. in height, at 2 per cent 0.45 in., at 3 per cent 0.8 in., at 3.5 per cent 1.2 in., and at 4 per cent more than 1.7 in., which is the height of the glass globe exposure. In testing, the lamp is raised slowly from a position near the floor to the ceiling of a working, as methane tends to accumulate near ceilings or in pockets where ventilation is deficient. Near the floor the methane content may be too low to give a visible gas cap, but as the lamp is raised the cap becomes visible and increases in height, sometimes giving a gas cap that almost fills the lamp interior. An improved form of Kohler safety lamp is provided with a bimetallic helix mounted in a stainless-steel tube. The free end of the helix is attached to a pointer rod, which casts a shadow

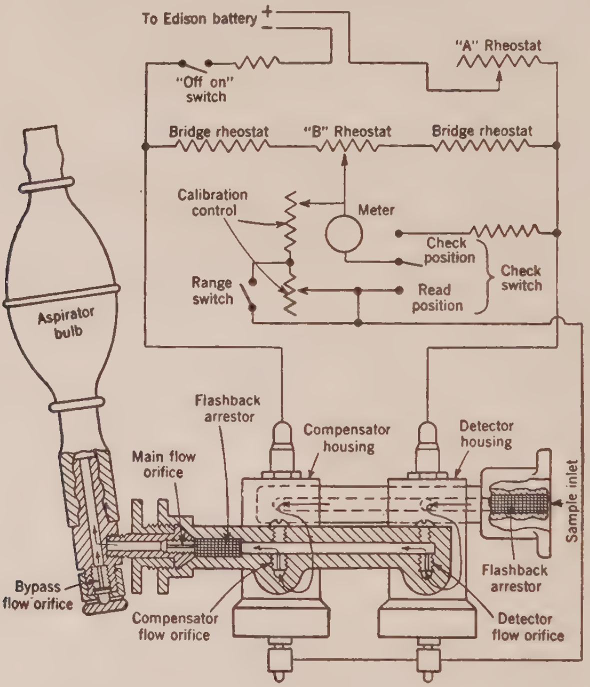

Fig. 123.—Methane detector, M.S.A.W.-8 type. (*U.S. Bur. Mines.*)

upon a scale indicating the approximate percentages of methane. The helix coils or uncoils in response to the heating by the gas cap. Gas-testing safety lamps must be clean and should be assembled and conditioned by experienced lampmen.

Several types of electrical methane detectors (Fig. 123) are available. They are satisfactory in quantitative indications, but all become inaccurate if used in atmospheres containing less than 16 per cent oxygen. These instruments are based on the principle that a heated coil of platinum shows increased electrical resistance as its temperature is increased. The atmosphere to be tested is drawn through a drying tube and past the platinum detector coil, which is heated by an electrical current to about 1000°C. When no combustible gas is present the electrical resistance of the coil remains constant, but where methane or other combustible gas is present the gas burns and raises the temperature of the coil, thus increasing its resistance, which is measured upon a compact Wheatstone bridge. Usually an Edison battery supplies current for the operation of the instrument. Various portable models are on the market, with dial markings designed to meet the specific needs of the user. For testing methane the dial readings indicate percentages of the gas, zero adjustment being made in pure air. These instruments can detect and measure extremely low percentages of methane.

Oxygen cannot be readily determined quantitatively except by gas-analysis methods. Where combustible gases are absent, the candle and acetylene lamp can be used to indicate oxygen deficiency. Normal oxygen content is 20.93 per cent; a vertical candle flame extinguishes at 17.6 per cent oxygen at sea level; the acetylene lamp flame will burn until the oxygen content reaches a low of 11.5 per cent. In both cases the character of the flame, its height and vigor, is influenced by the oxygen content. The safety lamp is the safest device to test for oxygen deficiency. The appearance of its flame, which is extinguished at 16.25 per cent oxygen, as compared with the flame in pure air, is a measure of the oxygen deficiency. Where combustible gases are suspected, all open-flame lights should be avoided and only a properly adjusted safety lamp used. Flash lamps and electric cap lamps are to be avoided when entering and testing suspected atmospheres unless a safety lamp, properly adjusted, is taken along.

Carbon Monoxide is detected by means of its reaction upon canaries or small animals and by the iodine pentoxide (hoolamite) detector. According to Bureau of Mines tests, canaries are more susceptible to carbon monoxide than small animals, but the response of individual birds is unreliable and uncertain. The iodine pentoxide detector is the most convenient and reliable indicator, but canaries are the best substitute where it cannot be obtained and entry is to be attempted. The U.S.

Bureau of Mines recommends that at least three birds be taken by exploring parties; the distress of any one bird may be taken as an indication of dangerous amounts of carbon monoxide. Birds will not indicate oxygen deficiency.

The iodine pentoxide detector (Fig. 124) makes use of a mixture of iodine pentoxide, fuming sulphuric acid, and granular pumice stone. In contact with carbon monoxide the white granules are changed from white to a succession of colors, bluish green of increasing depth, violet brown, and finally black, depending upon the concentration of carbon monoxide. The mixture is placed in small glass tubes tapered at the ends and hermetically sealed. The simple instrument in which the tube is placed, after both tips are broken off, consists of a metal barrel filled with activated charcoal through which air can be drawn by means of a rubber hand bulb and then forced through the tube containing the mixture. The bulb is squeezed 10 times in succession, forcing about 350 cc. of air through the mixture in the tube. A comparison tube in which five permanent colors are placed is mounted on the instrument alongside the tube through which the air to be tested has passed. These colors indicate percentages of carbon monoxide from 0.1 to 1 per cent. The inlet tube of the metal barrel has a corrugated tip to which a length of rubber hose can be attached, thus enabling a sample of gas to be drawn from a distance. The operator making tests should wear suitable respiratory apparatus for protection.

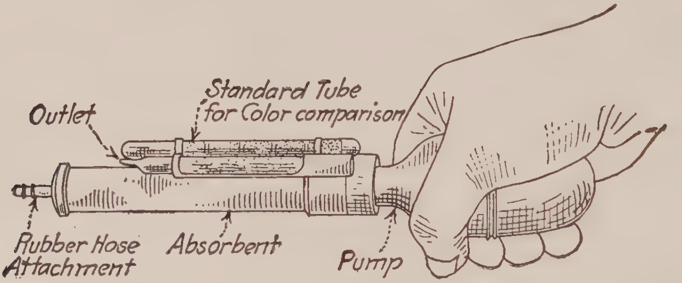

FIG. 124.—Carbon monoxide tester. (*Mine Safety Appliance Co. of America.*)

Hydrogen Sulphide gas is detected by its smell. A simple type of detector has been devised by the Bureau of Mines. Similar to the iodine pentoxide detector, it makes use of small tubes tapered and sealed at the ends and containing silver cyanide on activated alumina. Hydrogen sulphide gas turns the white granules to a dark-gray color. A suitable respiratory apparatus should protect the operator making the tests.

Production of Ventilating Currents.—Ventilating currents are produced by temperature differences, natural ventilation, or the use of mechanically operated fans, which may exhaust the air from the workings or may force fresh air into the mine, displacing and driving out the foul air.

Rock temperatures increase 1°F. for each 50 to 250 ft. of vertical depth. The difference between the temperature of the air within a mine and the outside air produces an air current, as is indicated in *a* and *b* of Fig. 125. Natural ventilation as it takes place in a metal mine is shown in the lower figure. Air warmed in the mine workings rises and escapes through raises and shafts, the outside air entering through other openings or shafts, and thus circulation takes place. The greater the temperature difference, the greater the amount of air in circulation. In very deep mines natural air currents may be surprisingly strong. However, the uncertainty as to the quantity of air and the difficulty of its control, as well as the seasonal variations in temperature, preclude reliance upon natural ventilation in all large and many small mines.

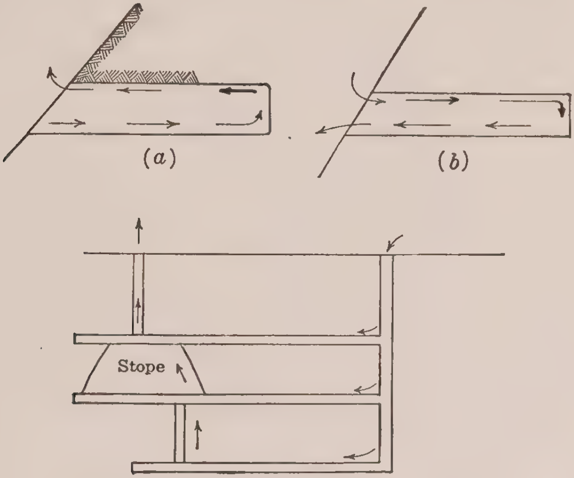

FIG. 125.—Natural ventilation: *a*, Winter conditions; *b* Summer conditions.

Very few examples of careful measurements of air currents circulated by temperature differences are available. Air volumes passing through the Tonopah mines in their prime were measured by B. O. Pickard. They ranged from 26,100 to 37,300 cu. ft. per min. as a result of natural ventilation. All were comparatively shallow and more or less interconnected. Years ago some measurements of air volumes were made on the Comstock mines and upcast air from four principal shafts open at that time showed a total volume of 235,835 cu. ft. per min. The incoming air of the Sutro tunnel measured over 27,000 cu. ft. per min. These were hot mines, but only limited work was being done.

Mechanical Ventilation.—Two main airways are necessary in mechanical ventilation, one for the incoming and the other for the outgoing current. In coal mines these airways may be entries or shafts and slopes; in metal mines they may be shafts, raises, and adits. In both cases the fan or blower may force air through one main airway, the other being open, or it may exhaust the air. Individual mine conditions determine whether a pressure or an exhaust system will be used. In a shaft mine a separate shaft is used as a main airway, since a pressure or exhaust system is difficult to apply to a shaft used for mine operation. In some cases, however, a shaft compartment may be bratticed off from the working compartments and used as a return airway.

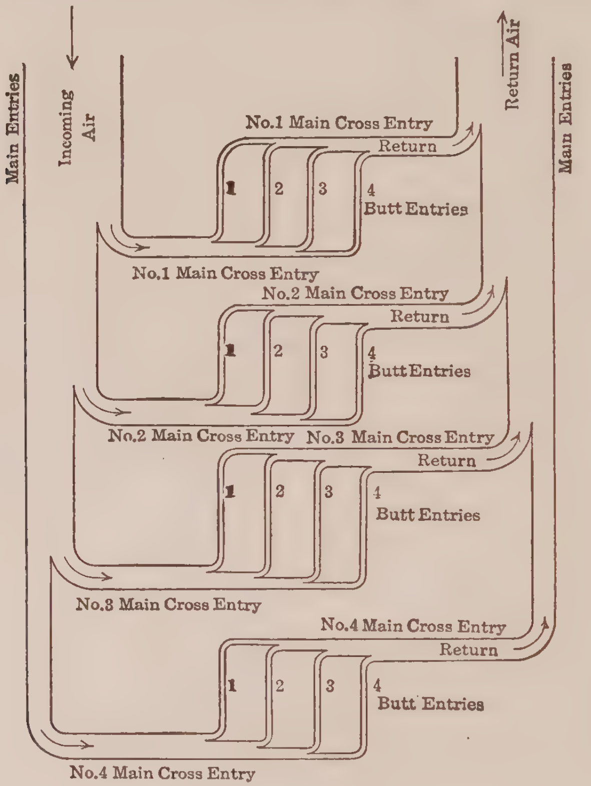

FIG. 126.—Ventilation chart of a coal mine.

Distribution and Control of Ventilating Currents.—In a metal mine the fan is generally used as an exhaust upon a system of raises connecting with the fan at the surface, the air coming in by all other shafts and openings; or the fan may be used underground, in which case air is drawn from one part of the mine and forced through raises to the surface. In other cases both underground and surface fans may be used.

In coal mines two or more parallel entries or slopes, separated by pillars of coal 40 to 50 ft. in width, provide the supply and exhaust airways. Figure 126 indicates the division or splitting of the incoming air between a number of cross and butt entries into the working places or rooms and the gathering together of the separate air currents in the main airway return. This general principle of air distribution is followed in mining operations. Certain accessories, described in the succeeding paragraphs, may be necessary.

Regulators.—Regulators are gates or doors at the entrance or exit of each split, designed to impose added resistance to the airway so as to prevent excessive air flows. The split of least resistance takes the most air unless a regulator is placed and adjusted to pass the required amount of air through the split or branch.

Break-throughs.—These are short connecting passages driven through the pillars as a pair of entries is advanced. They are cut through at distances of from 60 to 100 ft. and serve to return the air from one entry to the other. When the pair of entries has been advanced the permissible distance, another break-through is driven and the last one closed with a stopping, thus bringing the air to the working ends of the entries. In driving rooms in coal mining, break-throughs are driven at intervals, leading the air from room to room and keeping it close up to the working faces.

Stoppings are constructed of waste rock, concrete blocks, concrete, brick, or timber. Masonry stoppings are preferable, as they are freer from leakage. In a coal mine many stoppings have to be constructed and the cost involved is large. Systematic design and construction are a necessity both on account of the cost and the fact that the efficiency of the ventilation is dependent on the tightness of the stoppings.

Construction of Stoppings.—Stoppings are necessary at intervals of from 50 to 75 ft. along main and side entries. The crudest method of construction (Fig. 127*a*) is to pack a wall of waste 6 to 10 ft. thick in the opening. Clay is tamped in at the top, and after settlement has taken place more clay is rammed in and a fairly efficient stopping obtained. The cost of a stopping 6 ft. thick in one mine in District 7, Ill., was 5.4 cents per square foot of face. In another mine a stopping 12 ft. thick cost 7 cents per square foot, not including the transportation of the material. A tight wall constructed of shiplap boards and banked with dirt on either side is another method. Brick stoppings consisting of a single course of brick with a reinforcing pier at the center, as shown in Fig. 127*b*, are used at some mines. Concrete walls, concrete blocks, Pyrobar (bricks made of plaster of Paris), and timber are common forms of construction. Preference is given to brick, concrete, and concrete blocks. Gunite can be used to form an impervious layer over a stopping or to make airtight the joint between the structure and wall. The costs of different types as used in Illinois coal mines are given in Table 57.

TABLE 57

District	Type of construction	Total costs, cents per square foot
V	Concrete monolith	21.6
VI	Concrete monolith	11.4
VI	Concrete monolith	15.0
VI	Brick coated with cement	16.6
VII	Concrete blocks	10.6
VIII	Concrete monolith	25.1

Overcasts and Undercasts.—These are necessary where two airways meet at a right angle in the same plane. Both serve the same function, although in practice the overcast is preferred. The overcast may be likened to a bridge. It is constructed by blocking the ends of the cross airway with stoppings, constructing a U-shaped passageway above the other airway, as shown in Fig. 128, and bridging the stoppings with an airtight floor. Overcasts are constructed of timber, timber and steel, brick, monolithic concrete, cement blocks, steel and concrete, Pyrobar, and brick and timber.

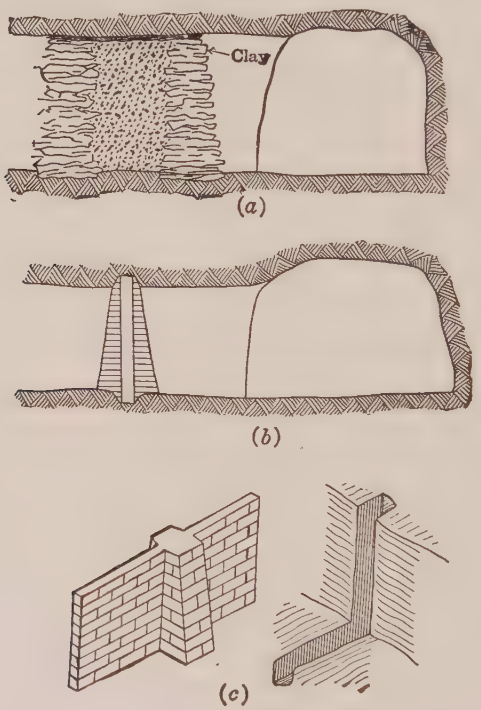

FIG. 127.—Stoppings.

Doors are necessary at many points to prevent short-circuiting of the air current during the passage of cars and locomotives. They are usually constructed in pairs so that in principle they act as an air lock. The distance between the doors is sufficient to accommodate a train of cars and locomotive. They are arranged to open against the air current. In many cases heavy wooden doors are used.

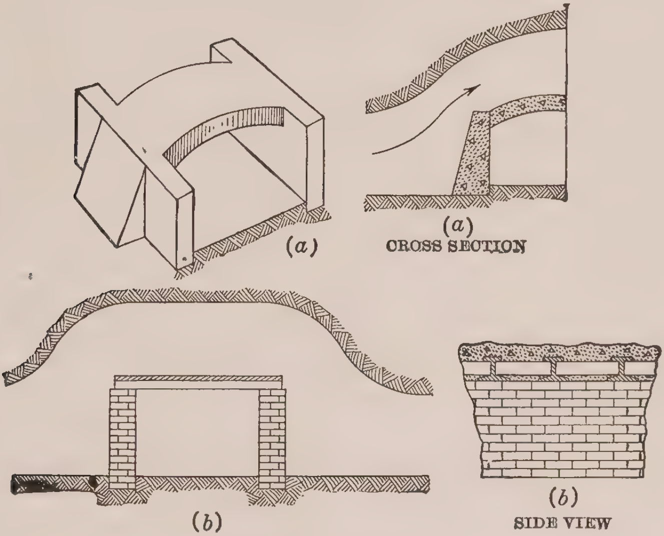

Fig. 128.—Overcast.

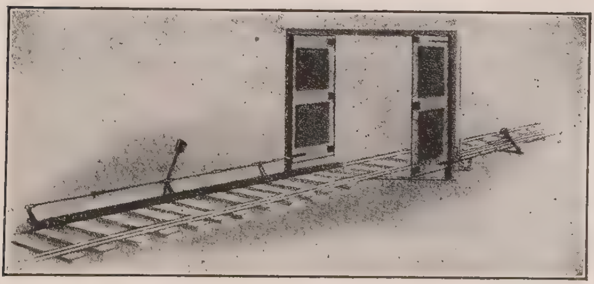

Fig. 129.—Canton mine door.

The Canton mine door shown in Fig. 129 has many advantageous features. It consists of a pair of light metal doors placed at an angle to the center line of the haulage way. The doors can be opened by a lever which is operated by hand or by the trip of cars. The doors open in opposite directions.

Explosion Doors.—Explosion doors are sometimes constructed in stoppings with the object of preventing the blowing down of the stopping.

It is evident that the lighter forms of construction are liable to destruction in the case of an explosion. Stoppings must be repaired where destroyed in order to carry the air into the workings, and much delay may ensue. Explosion doors therefore serve to protect the stoppings, and their use may enable a mine to be quickly and safely entered after an explosion. At one Illinois mine it is the practice to place a door, 4 ft. 3 in. wide and of a height equal to the height of the stopping, in every eighth stopping. The door is constructed of two thicknesses of 1-in. lumber, provided with an iron rod at the top, and is lightly fastened at the bottom.

Brattices are used where it is necessary to ventilate dead ends. Single entries and rooms driven considerable distances beyond break-throughs require the use of brattices for proper ventilation. The brattice is a partition constructed of cloth and dividing the entry or room into two parts lengthwise of the working. A line of props is necessary in order to carry the cloth. Air passes up one side of the partition and down the other. Brattice cloth, of which many types are available, consists of canvas or tightly woven burlap treated with chemicals (ammonium phosphate) or some composition to render it fireproof.

Air Pipes.—Air pipes, which are also used for the ventilation of dead ends, in some respects are more convenient than brattices. Air pipe is constructed of galvanized iron in 12-ft. lengths, joined by bell-and-spigot joints. The longitudinal seams are riveted and soldered. The joints are wrapped with several layers of tarred burlap held in place by rope or wire. The diameters used are 6, 8, 11, 15, and 19 in. An air pipe is suspended by light U-shaped hangers, the ends of which are driven into timbers. A hanger is placed at the center and at the end of each section. In some cases ropes are used. Usually power-driven fans furnish the air to the air pipe. Canvas pipe rendered impervious to air by a special composition has been found convenient for directing air currents in metal mines and avoids some of the objections to cloth brattices. Ventilation of shafts during construction is effected by air pipes or by a wooden partition or brattice dividing one compartment from the other.

Curtains are used where temporary stoppings are required. They are constructed of canvas or brattice cloth. They afford at best only a partial barrier to the air current. In long-wall mining, gates are partially closed by a half curtain. This prevents too large a volume of air from passing through the gate.

Tunnels and Adits.—Ventilation of tunnels and adits during construction is effected by air pipes or by shafts that connect with the tunnel at intervals. It is obvious that the latter method can be used only where the cover is of nominal thickness. In prospecting work various expedients are resorted to to obviate the use of power.

Velocity of Air Currents.—The velocity of air currents in airways is restricted. In main airways velocities of 1,000 ft. per min. or greater are permissible, while in splits the greatest velocity should not exceed 600 to 700 ft. per min. The Pennsylvania mining law (anthracite) limits the velocity of air to a maximum of 450 ft. per min. in each split.

Control of Ventilation.—The control of ventilation is best effected by constructing a ventilating chart showing the main air currents and first and second splits. The length and cross section of each split and the quantity of air are shown on the chart. At regular intervals the quantity of air in both main and subordinate airways is determined by anemometer measurement, and where changes are required doors and regulators should be readjusted. Tests of the quality of the air in different parts of the mine should also be made at intervals.

Barometric observations are regularly taken at coal mines. They are especially necessary at gaseous mines, since periods of low barometric pressure have sometimes been found to coincide with excessive outpourings of gas. A low barometer indicates danger.

In dry and dusty coal mines daily observations of underground humidity are necessary, and whenever the relative humidity falls below a safe minimum immediate steps should be taken to charge the air currents with moisture. A dusty mine can be kept safe by wetting down the coal dust at frequent intervals and preventing the dust from drying out by keeping the air current near saturation.[1]

In metal mines conditions are less critical, and less detailed attention is given to the control of ventilation. Nevertheless it is an important feature, and mine foremen should be required to make the necessary observations from time to time. The ventilation system should come up for general review and study by the superintendent at least once a year. In all metal mines air pipe and fans should be provided for temporary use in ventilating dead ends while connection is being made through raises and winzes.

Formulas for the Flow of Air through Workings.—Mine ventilation formulas stem from the work of J. J. Atkinson (1854) and from the study of the flow of fluids through pipes. Their use enables approximations to be made relative to quantities, pressures, power, and other factors. The following symbols are used.

P = pressure, pounds per square foot of net section of airway.
a = net area of airway, square feet.
i = inches of water pressure above atmospheric pressure as measured by a manometer.
Q = quantity of air flow, cubic feet per minute.
V = velocity, feet per minute. (Average over section.)

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[1] See *Bull.* 83, *U.S. Bur. Mines*, for discussion of humidity in coal mines.

K = friction factor at standard air density of 0.075 lb. per cu. ft.; for other densities multiply by the ratio of observed air density and standard air density. It is the pressure in pounds required to overcome the friction caused by 1 sq. ft. of rubbing surface at unit velocity.

S = inner surface of airway, or length by perimeter; square feet.

Hp = horsepower for given quantity and pressure.

P_s = static pressure.

P_v = velocity pressure.

The value of K at standard density is summarized from various authorities as follows.

For galvanized iron piping: 0.000,000,0016.

For canvas piping: 0.000,000,0022.

For air passages that are concreted, wood-lined, or of exceptional smoothness, untimbered, straight, regular, and free from obstructions: 0.000,000,0036.

For air passages that are untimbered, fairly smooth, straight, unobstructed, and regular in size: 0.000,000,0073.

For air passages that are untimbered and moderately crooked, irregular, obstructed, and rough: 0.000,000,0109.

For air passages that are timbered but otherwise fairly smooth, straight, unobstructed, and regular in size: 0.000,000,0146.

For air passages that are timbered and moderately crooked, irregular, obstructed, and rough: 0.000,000,0182.

For air passages that are timbered and very crooked, irregular, and rough: 0.000,000,0219.

The following formulas are generally used:

$$P = \frac{KSV^2}{a} = \frac{KSQ^2}{a^3}$$

$$Q = Va = \sqrt{\frac{Pa^3}{KS}}$$

$$P = 5.2i$$

Velocity pressure in inches of water gage

$= 0.000,000,0623\,V^2$ for standard pressure.

$P_s + P_v$ = total pressure to overcome friction and velocity $= P_t$.

$$Hp = \frac{P_t a V}{33,000}.$$

Where two or more airways branch from a given point, the quantity of air taken by each branch (Q_1, Q_2, Q_3) from the total quantity delivered at the branching is given by the following formula:

$$Q_1 = \frac{Qa_1\sqrt{\frac{a_1}{K_1S_1}}}{a_1\sqrt{\frac{a_1}{K_1S_1}} + a_2\sqrt{\frac{a_2}{K_2S_2}} + a_3\sqrt{\frac{a_3}{K_3S_3}}}$$

Similar formulas may be written for Q_2 and Q_3. The factor $a\sqrt{\frac{a}{KS}}$ is sometimes called the pressure potential of the split. Thus the amount of air taken by a given split is the total quantity multiplied by the ratio of the pressure potential of the split and the sum of all the pressure potentials of all the splits originating at the given point. The pressure at the entrance of each split is the same.

W. D. Sheldon, Jr., in reference to parallel airways, defines the specific capacity of each airway as the ratio of the quantity flowing and the square root of the inches of water gage necessary to overcome the resistance of the particular airway and the quantity assumed for the airway; in other words, the capacity of the airway per inch of water gage. The capacity of each airway is the ratio of its specific capacity and the sum of all the specific capacities of the parallel airways multiplied by the total quantity of air to be divided. The resistance of the multiple airways in inches of water gage is the square of the total quantity of air divided by the square of the sum of the specific capacities of the airways.

The application of the equation for splitting air currents is exemplified by the following. Assume that three airways receive their supply of air from a main airway carrying 50,000 cu. ft. per min. The dimensions of the airways are as follows: (*A*) 10 by 6 by 1,000 ft. long; (*B*) 4 by 6 by 300 ft. long; (*C*) 12 by 8 by 2,000 ft. long.

The value of $a\sqrt{\frac{a}{S}}$ for each airway is (*A*) 3.1333; (*B*) 1.5178; (*C*) 3.3255. The sum is 7.9766. The amount of air in each airway is given by the product of the respective ratios $\frac{3.1333}{7.9766}$, $\frac{1.5178}{7.9766}$, $\frac{3.3255}{7.9766}$, and the total quantity of 50,000 cu. ft. The percentage division is, respectively, 39.3, 19, and 41.7. The quantities are, respectively, 19,650, 9,500, and 20,850 cu. ft. per min.

Resistance of Mine.—The simplest mine development provides for an incoming airway, an intervening area in which the air supply is divided into one or more airways (more or less parallel when more than one), and an outgoing airway that collects the air and discharges it into the atmosphere. Where the length and other conditions of the airways can be determined, the resistance of the mine may be approximated by determining the respective resistances of the incoming and outgoing airways and the resistance of the parallel or net of airways connecting these main airways. To facilitate this calculation, the probable quantity of air that may be required by the mine may be assumed.

Each mine is a separate problem. The factors that enter into the picture are underground temperatures and humidities at working places, the number of men employed, the amount of blasting, the amount and kind of ore movement, and the dust problem. In coal mines the liberation of methane and other gases from the coal must be considered. Conditioning of air, lowering underground temperatures, increasing or decreasing humidity, and diluting dust-laden air are factors having to do with the quantity of air in circulation. On the other hand, the pressure required to force air through a mine varies as the square of the quantity as well as the square of the velocity, while the power varies directly as the product of pressure and quantity. Permissible velocities vary and may be relatively higher in main airways than in working places. To double the quantity of air flowing in a given mine require 4 times the pressure. Very deep, large mines require large quantities of air in circulation, and where high temperatures are encountered still larger quantities are necessary. The ventilation layout requires good engineering wherever high efficiency is desired.

Types of Fans.—Primary units or fans, for serving the whole or a considerable part of a mine, evolved in an interesting way. Coal mining first felt the need for controlled ventilation. Ventilating furnaces, long used in English coal mines, were supplanted by power-driven fans and blowers. One of the earliest of these was the Root blower, which still survives and is useful where high pressures are necessary. Early use of the propeller fan did not survive, and the large-diameter slow-speed centrifugal fan such as the Guibal and similar types became dominant. These fans were used as exhausters and were provided with a flared upcast and a direct-connected steam engine. Various designs, some of smaller diameter and higher speed, were developed and finally resulted in the modern centrifugal fan, which has been widely used for both coal- and metal-mine ventilation.

The centrifugal fan has many shallow blades close to the periphery of the rotor. Forward-tipped and backward-tipped blade rotors more or less supplanted radial and other blade designs. Higher speeds, smaller diameters, and better efficiencies were obtained with these fans, of which many are in use. The forward-tipped blade type requires about 3,000 ft. per min. peripheral speed and the backward-tipped type about 5,000 ft. per min. to develop 1-in. water gage. A rapidly rising power curve characterizes the forward-curved type as the pressure diminishes and the quantity of air delivered increases; the full backward-curved type gives a power curve that reaches a maximum and then diminishes as static pressure declines and quantity increases. The latter is called a nonoverloading fan, while the former requires an oversize motor for its drive.

The disk fan is the pioneer of the propeller-type fan. It has six broad-ended blades within a short cylindrical housing. Peripheral speed in the largest fans was about 7,000 to 10,000 ft. per min. The pressure characteristic was low, some 1.5-in. water gage. It was low in cost and easily installed, and could be direct-driven by motor. Its efficiency was low. Many fans of this kind were installed in small mines where mine resistance was low.

The inception of the modern propeller fan was probably an outgrowth of the experiments made by F. A. Steart in 1923. He tried out two-bladed airplane propellers mounted in series upon a shaft driven by a motor. A mechanical efficiency of some 70 per cent and a pressure of 3.5-in. water gage was obtained with six propellers mounted upon a common shaft. His work attracted attention, and subsequently various designs of multi-blade propeller fans appeared to which the general term "axial-flow fan" has been applied.

As an example of the simplest axial-flow fan, the Aerodyne Junior consists of a 12-bladed rotor 3 to 10 ft. in diameter, blades narrow-tipped, in a short cylindrical housing that also supports the shaft bearings. An overhung pulley provides for a V-belt drive. Pressure is up to 2-in. water gage. A two-stage arrangement, which develops pressures up to 4-in. water gage, consists of two rotors with stationary diffusion plates between them. Another example is the 8H Aerodyne, which consists of an eight-bladed rotor within a cylindrical housing. Guide vanes are behind the rotor. The ellipsoidal, stream-lined rotor shell closes out the hub section, preventing backflows and shaping the discharge current. Hub adjustments vary the pitch of the blades, which are curved and rectangular in shape. This type is called a vane axial-flow fan. Maximum peripheral speeds of such a fan are 25,000 ft. per min., and maximum pressures are about 8-in. water gage or more.

Maximum static efficiency of the foregoing axial-flow fans depends upon the pitch of the blades and the specific volume of the fan. Specific volume is the quantity of air in cubic feet per minute divided by the product of the square of the fan diameter and the square root of the pressure in inches of water gage for air of 0.075 lb. per cu. ft. Maximum static efficiency is approximately 75 per cent for a specific volume range of 1,400 to 2,200 cu. ft. per min. Below and above these limits static efficiency diminishes.

Nonoverloading characteristic, a wide range of quantities and pressures for a single unit, simplicity of construction, and low cost of installation are important advantages of the axial-flow type of fan. Ventilating currents may be reversed by changing the direction of rotation.

Secondary, supplementary, or booster service is supplied by smaller fan units installed at various points underground and drawing their air supply from the main ventilating currents. Usually centrifugal fans of the forward-tipped blade type are used for this service where Ventube or ventilating pipe may be required to ventilate dead ends. They are driven by direct-connected motors. Compact, light-weight propeller type blowers such as the Coppus and similar ones are widely used because

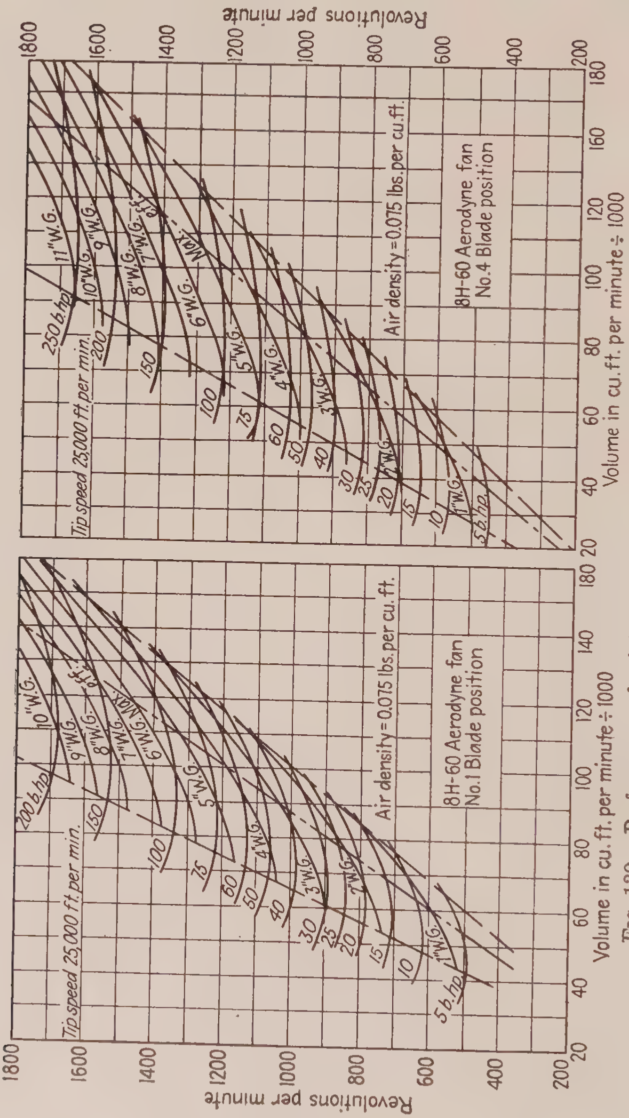


FIG. 130.—Performance of axial-flow fan, 60-in. diameter. (*Jeffrey Mfg. Co.*)

they are readily portable and efficient. Small axial-flow fans are also being used for booster service in high-resistance airways.

The high-pressure blower, such as the Root, is used for the ventilating of long tunnels under construction. Pressure is measured in pounds rather than inches of water gage. Conduits are often of great length. They may be used as either exhausters or blowers and are sometimes driven by variable-speed motors.

Fan Rating.—Centrifugal fans are given rating curves by the manufacturer. The rating curve is determined by connecting the fan discharge to a short duct of the same area as that of the fan discharge and operating the fan at its designed speed. Pressure, quantity, and power readings are made for the fan with the duct completely closed and at various fractions of opening, up to full opening. From these readings, which are plotted to show the relation between quantity and pressure, quantity and power, the mechanical efficiency of the fan may be calculated. For a given speed there is a corresponding quantity and pressure. For a constant duct opening and air density, the quantity discharged varies directly as the speed or r.p.m. of the fan wheel; pressure varies as the square, and power varies as its cube; but the mechanical efficiency remains constant. Thus, if the rating for a fan is known for a given

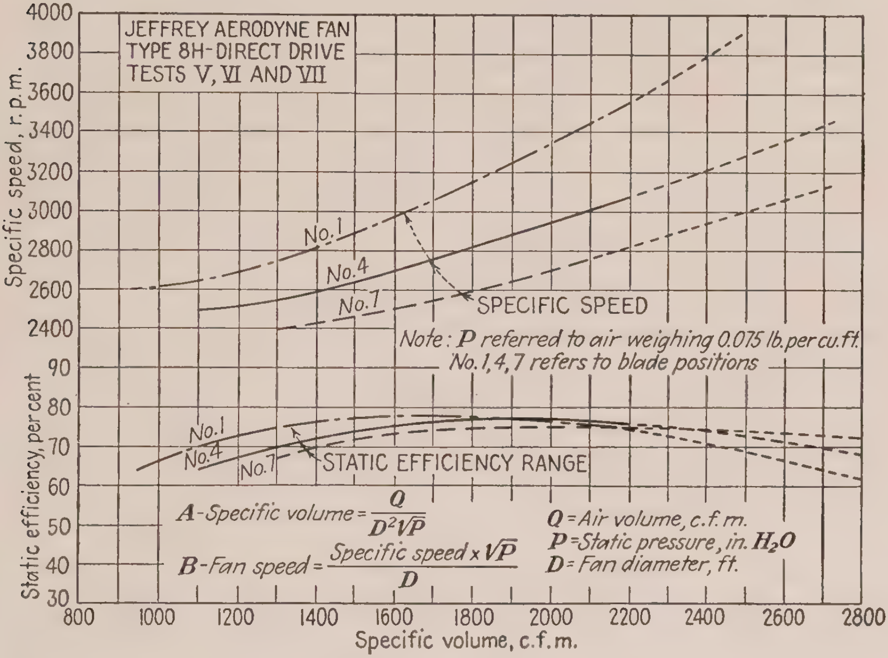
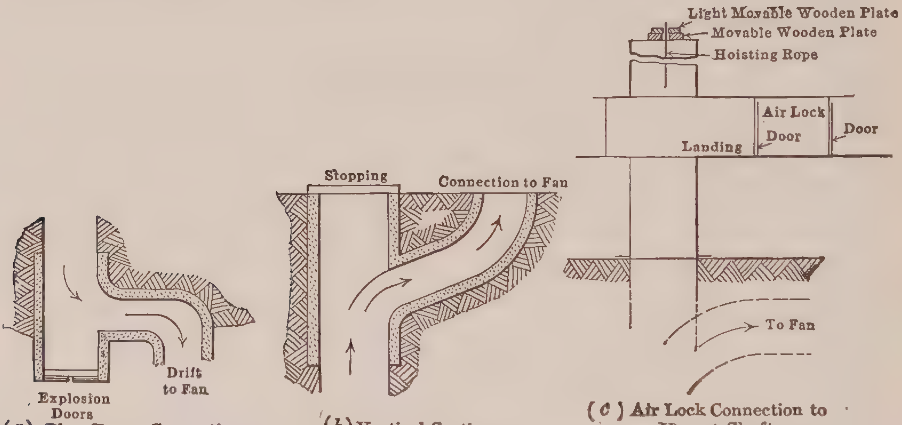


FIG. 130*a*.—Relation between specific volume, fan speed, and static efficiency for various blade positions. (*Jeffrey Mfg. Co.*)

speed, its corresponding quantity, pressure, and power can be calculated for other speeds. The upper limit of speed for a given fan is determined by its mechanical design, the strength of the materials used in its construction, and the factor of safety assumed for the conditions of maximum output. Figure 130 illustrates rating curves for an axial-flow fan. Figure 130*a* indicates relation between specific volume, fan speed, and static efficiencies for axial-flow fan.

Three terms used in describing fan performance require definition. Static pressure, measured close to the fan, is the pressure against the walls of a duct into which a stream of air is forced; velocity pressure is the pressure required to produce a given velocity in the duct; and total pressure is static pressure plus velocity pressure. In determining the static pressure of a pressure fan the connecting tube of the manometer is placed flush with the walls of the duct while total pressure requires that the inlet tube of the manometer extend into the duct and face the direction of the air current; the difference between the total and static pressures is the velocity pressure. Fan efficiencies are often based on static pressure. The static pressure of an exhaust fan is negative or below the atmospheric pressure, and the corresponding total pressure is less than the static pressure by an amount equal to the velocity pressure. Centrifugal fans may be expected to give mechanical efficiencies of 60 to 75 per cent and propeller fans up to 80 per cent or better in well-designed units. Tapered discharge sections are a feature of axial-flow or propeller fans; guide vanes also control turbulence.



(*a*) Plan Entry Connection to Fan

(*b*) Vertical Section Upcast Shaft Connection

(*c*) Air Lock Connection to Upcast Shaft (Vertical Section)

FIG. 131.—Ventilating fan connections.

Surface Arrangements of Fans.—Three methods of connecting fans with mine workings are shown in Fig. 131. The first of these, *a*, is the usual method of connecting a fan with the ventilating entry where connection is made with the surface. The entry is extended to the surface and terminates in a pair of explosion doors. At right angles to the entry a connection is made to the fan drift. The airways are supported by concrete. The arrangement can be used with either pressure or exhaust fans. A connection with an upcast shaft is illustrated in *b*. In *c* is shown an arrangement where the upcast shaft is used as a working shaft. The top of the shaft is connected with a steel housing. On the landing level an air lock, constructed of steel and of sufficient size to accommodate several cars, is connected with the housing. Steel doors are used. Small hand-operated valves are placed in the doors so that the air pressure can be equalized on either side of a door, thus allowing each door to be opened easily. The hoisting rope passes through a heavy wooden plate, on the top of which is a loose wooden plate. The lateral vibration of the rope swings the latter from side to side. The double plate makes a moderately efficient joint. In some shaft arrangements air locks are provided on each side of the landing housing. An extreme arrangement for an upcast shaft consists in the construction of the shaft house in such a manner as to permit the entire landing floor to be placed under the vacuum of the fan. Cars can be caged and removed from the cage without the use of air locks. Steel and masonry construction is necessary to accomplish this, and the arrangement is on the whole too costly to be used except under unusual conditions. Figure 132 illustrates the installation of an axial-flow fan at the Moose shaft in Butte.

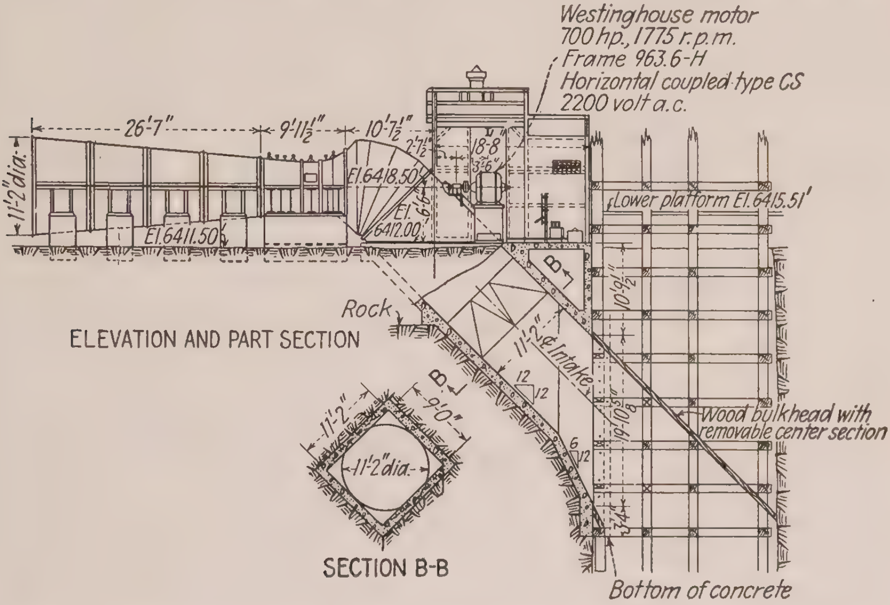


FIG. 132.—Axial-flow fan and details of installation, Moose shaft, Butte. (*A.I.M.E.*)

Ventilation of Metal Mines.—During exploratory operations mechanical ventilation of dead ends is essential. For this purpose small fans or blowers and air pipe or canvas piping are generally used. The modern canvas duct has displaced the metal air pipe, as it can be more conveniently used and its flexibility is an asset. Compressed-air-operated blowers of the Coppus type are widely employed where electric power is not available underground. They are far more reliable than exhaust from air drills and give dependable ventilation. Injector blowers are also used but are not so satisfactory or economical as the mechanical blowers.

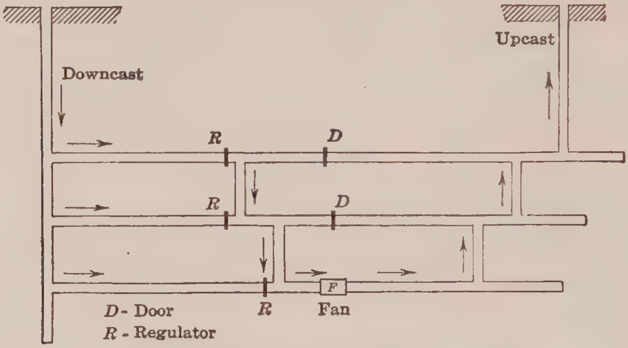


FIG. 133.—Fan ventilation, longitudinal section.

When secondary outlets are holed through to the surface, natural ventilation begins and may suffice for much of the mine area; as stopes connect levels more air circulates, and relatively small amounts of air are required in exploratory operations. As development levels are extended, natural ventilation may be enhanced by generous cross sections, by special care in planning inlet and outlet airways, and by a general plan. Auxiliary power ventilation is always necessary for dead ends such as drifts, crosscuts, winzes, and raises.

In shaft mines, inlet and outlet shafts or an inlet shaft and an outlet to the surface through a series of raises are a minimum requirement. An exhaust fan connects with the outlet. Fans are sometimes placed underground. Figures 133 and 134 show two generalized plans. The ventilation plan for large mines may be very complex. Blasting fumes are taken care of at points of blasting by water blasts or sprays and by opening compressed-air valves near the face. Dust is diluted by air blown through flexible pipe at the face, by wet drilling, and by wetting down by water sprays. Dust at loading chutes is minimized by wetting down, by water sprays, by ventilating currents, or by removal by exhaust fans to filtering units or spray chambers.

In the mining of large bodies of sulphide ore difficulty has arisen over the frequent occurrence of mine fires that originate in the mined-out periphery of the ore body and are exceedingly difficult to control or extinguish. Fire zones are bulkheaded and the sulphurous smoke is sealed off. In spite of this precaution, sulphur smoke sometimes penetrates the workings and interferes seriously with operations. At present the practice is to surround fire zones with air under moderate pressure, which causes an inflow of air to the fire zone and effectually prevents the sulphur smoke from reaching the workings. This is accomplished by pressure fans and by making incasts of the connecting passages leading to the fire zone.[1] The high temperatures that often characterize the workings near fire zones are reduced by forcing larger volumes of air through them than can be obtained by natural ventilation.[2]

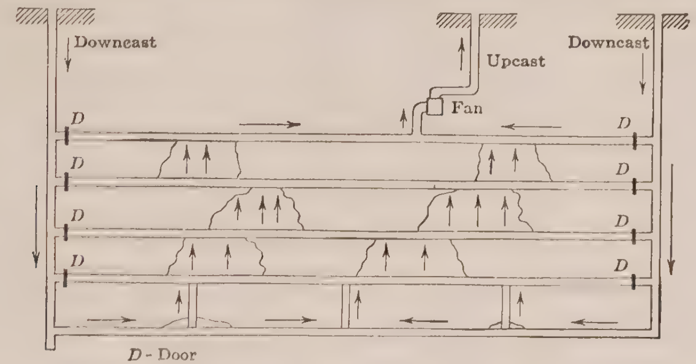


FIG. 134.—Fan ventilation, longitudinal section.

Ventilation of Coal Mines.—The distribution of the air currents in long-wall workings is illustrated by Fig. 135, which is taken from *Bulletin* 2, *Illinois Coal Mining Investigations.* The example is typical. Figure 136 illustrates the distribution of air in room-and-pillar workings. The ventilation of a coal mine, unlike that of a metal mine, is more or less systematically planned before the mine is opened. Provision is made for upcast and downcast shafts and airways. In the chapter on Development several mine plans are given and serve to illustrate the systems in use. The quantity of air is determined by the probable maximum daily output of the mine; the permissible velocities and the length of the airways can be taken from the preliminary development plan, and the cross sections of main and side entries are determined. The probable resistance of the mine is next calculated by adding the resistances of inlet and outlet airways and the resistance of one or more side entries.

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[1] Fire Fighting Methods at Mountain View Mine. *Trans. A.I.M.E.*, vol. 52, p. 534.

[2] Ventilation of the Copper Queen Mine. *Trans. A.I.M.E.*, vol. 52, p. 508.

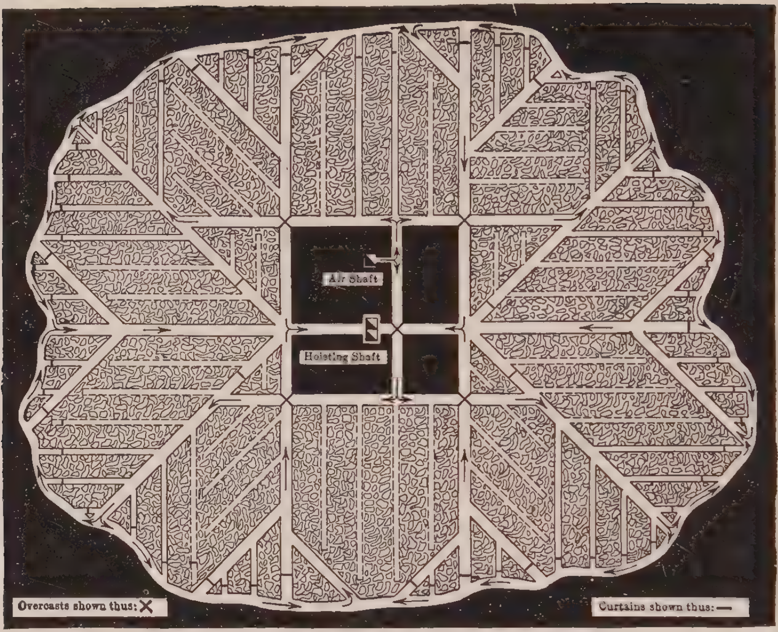

FIG. 135.—Plan of ventilating system used in long-wall coal mining. (*Illinois Coal Mining Investigations.*)

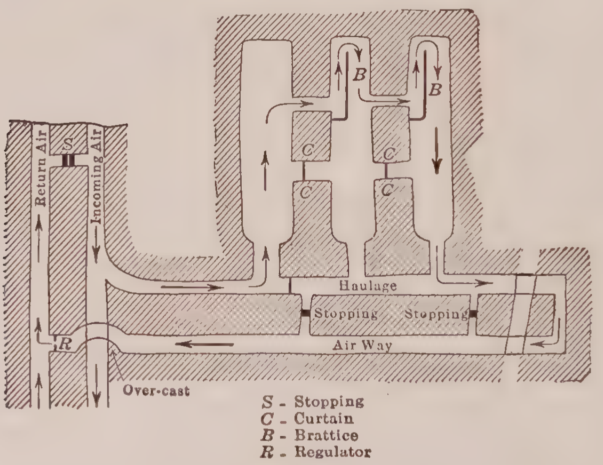

FIG. 136.—Plan of ventilating system used in room-and-pillar coal mining.

Since the side entries are ventilated by parallel air currents that are fractional parts of the main air current, their resistances are not cumulative. The two factors, quantity of air and mine resistance, determine the size and type of fan required.

The proportioning of airways is an important feature. The necessity of avoiding excessive velocities and high resistances by the use of airways of ample cross section is illustrated by Tables 58 and 59. The pressure and horsepower required have been calculated for an airway 1,000 ft. in length and for different quantities of air. For purposes of comparison, an airway of twice the cross section and carrying 100,000 cu. ft. per min.

TABLE 58.—LENGTH OF AIRWAY 1,000 FT. ($K = 0.00000001$)

Dimensions	Area, sq. ft.	Velocity, ft. per min.	Vol., cu. ft. per min.	Water gage, in.	Hp., air	Hp., 50 per cent efficiency
8 × 6.25	50	500	25,000	0.274	1.08	2.16
8 × 6.25	50	750	37,500	0.617	3.63	7.26
8 × 6.25	50	1,000	50,000	1.100	8.61	17.22
8 × 6.25	50	2,000	100,000	4.385	68.84	137.68
8 × 12.5	100	1,000	100,000	0.788	12.37	24.74

TABLE 59

Cu. ft. per min.	Cost per hp.-hr.				Cost, 24 hr. at 1 ct. per hp.-hr.	Cost per 1,000 cu. ft. per 24 hr.
	0.5	1.0	2	3		
25,000	$.0108	$ .0216	$ .0432	$ .0648	$0.5184	$0.0207
37,500	.0363	.0726	.1452	.2178	1.7424	0.0464
50,000	.0861	.1722	.3444	.5166	4.1328	0.0826
100,000	.6824	1.3768	2.7536	4.1304	33.043	0.3304
100,000	.1247	.2478	.4948	.7422	5.938	0.0593

has been calculated. In the second table power costs have been calculated, as well as the cost per unit of 1,000 cu. ft. per 24 hr. The comparison is obvious. The last two examples raise the question of comparative costs of construction. The airway of 100-sq. ft. cross section may be expected to cost somewhat less than twice as much as the 50-sq. ft. cross section. If we assume a cost of construction of 10 cents per cubic foot and the cost of the larger airway as twice that of the smaller, the respective costs are $10,000 and $5,000 per 1,000 ft. Assuming a year of 250 days, the respective power costs for the airways are $1,484.50 and $8,260 (power at 1 cent per horsepower-hour). The larger airway may be expected to cost more for annual maintenance, but this would not be sufficient to add more than a nominal sum to the power cost. The differential in power costs in favor of the larger airway greatly overbalances the increased cost of construction. An extreme example has been taken in order to point out the necessity of comparing construction, maintenance, and power costs for airways of different cross section.

Gob fires, sometimes encountered in long-wall workings, are caused by a combination of factors, such as fine particles of coal, finely divided iron pyrites, moisture, air confined in the interstices of the gob, and roof pressure. They are controlled by bulkheads constructed about the fire zone and by pressure ventilation.

Cost of Ventilation.—The cost of ventilation is a small charge against mining when compared with other cost items. Included are power, supplies, labor, and the repairs of airways. Supplies and labor are nominal in amount. Where motor drives are used, attendance is reduced by distant control and by thermal couples at important bearings, which operate relays sounding alarms when unsafe conditions arise. The maintenance of airways is a variable item. In many coal mines the auxiliary airways are timbered only where needed, and only enough work is done upon them to keep them open. Roof falls are allowed to remain where they do not make too serious an obstruction. As main airways are substantially timbered, their maintenance is seldom costly. By the use of treated timber, maintenance is reduced to a minimum figure. In large metal and coal mines a ventilating engineer is given charge of the ventilating system and provided with a sufficient number of assistants to inspect and to control the mine ventilation.

A summary of mine ventilation costs, as given by G. E. McElroy, shows a range of 0.8 to 11 cents per 100,000 cu. ft. of air; cubic feet of air per minute per horsepower ranged from 390 to 1,440 cu. ft. for main fans; total air circulated ranged from 40,000 to 350,000 cu. ft. per min.; mean yearly costs ranged from $11,656 to $97,899; costs per ton of ore from 0.32 to 18.7 cents.

In *Technical Publication* 979, A.I.M.E., C. B. Foraker gives details of the ventilation costs of the Magma mine (Arizona). The average for 6 years was computed as follows: Labor $9,078, power $17,339, and supplies $7,823, or a total of $34,240. The unit costs per ton of ore ranged from 0.115 to 0.216 cents and per pound of copper from 0.00088 to 0.0014 cents. Of the total ventilation costs labor constituted 26.51 per cent, power 50.64 per cent, and supplies 22.85 per cent. An air-conditioning plant installed at Magma required 494,310 kw.-hr. for its operation during the month of November, 1937, or 15.93 kw.-hr. per ton of ore mined. Exclusive of power the cost of operating the plant was $685.74 for labor and $17.33 for supplies; cost per ton of ore mined was 2.26 cents.

Air Conditioning.—High humidity conditions prevail in most mines, and where temperatures exceed 90°F. efforts are made to control the humidity of the working places by conditioning the air. An increased quantity of air was the first expedient. At the Champion Reef mine (India) air flow was increased to 600,000 cu. ft. per min., or 13 tons of air per ton of ore, but precooling of the inflowing air became necessary; in the Rand mines (South Africa) air quantities in the very deep hot mines approximated 10 tons per ton of ore, but in the deepest mines precooling became necessary; in 1922 ventilation at the Ooregum mine (India) was increased from 80,000 to 200,000 cu. ft. per min., or 20 tons of air per ton of ore, but here also precooling finally became necessary. Mines exceeding 5,000 ft. in depth and mines in formations where the temperature gradient is high are confronted with the necessity of precooling and dehumidifying. In the accompanying table are given data of important mines where refrigerating equipment has been installed. The number of such mines is small.

DEEP MINES HAVING REFRIGERATING EQUIPMENT FOR PRECOOLING AIR

Mine	Date	Depth, ft.	Rock temp., °F.	Working temperature, °F.		
				Wet bulb	Dry bulb	
Morro Velho (Brazil)	1920	8,050	130	81	90–126	*S*
	1929	...	...	...	...	*U*
	1939	...	...	...	...	*S*
Randfontein Estates (S. Afr.)	1932	...	...	...	...	*U*
Robinson Deep	1934	8,500	103	82	...	*S*
Chris Shaft	1941	...	...	...	...	*S*
Butte (U.S.)	1936	4,100–4,400	120	...	...	*S-U*
Magma (U.S.)	1937	4,400–4,600	150	78.5	88.5	*U*
Ooregum (India)	1938	8,500	134	73–83	97–117	*S*
Champion Reef (India)	1940	8,500	134	...	...	*S*

NOTE.—*S* = refrigerating plant at surface; *U* = refrigerating plant underground. At the Morro Velho a refrigerating plant was installed in 1929 on the 6,000-ft. level but was removed to the surface in 1939.

A. Caplan states that in the Kolar deep mines a dry-bulb temperature as high as 120°F. with wet-bulb temperatures not higher than 85°F. and air velocity not exceeding 600 ft. per min. can be withstood by miners. Circulation of precooled or dehumidified air lowers both dry-bulb and wet-bulb temperatures and may give control over working temperatures. Most of the refrigeration installations are placed at the surface because cooling water is readily available there and also because at great depths circulation of cooling water from the surface is not practicable. In the Butte mines cooling water is circulated from the surface to the 3,600-ft. level and to the air-cooling units at the deep levels. It is then returned to the surface, where its temperature is lowered in cooling towers. At Magma underground water is available for cooling and the refrigerating units are placed underground. Placing the units close to the hot zones of the mine is favored where cooling water is available or can be circulated by pumping.

In extremely cold conditions it is sometimes necessary to heat air currents entering the mine so as to prevent the formation of ice in shafts and entries. This is done by turning steam into the air current and by providing heating radiation in the airway. In some coal mines where very dry conditions prevail air sprays are operated so as to increase air humidities and diminish the hazards from coal dust; steam is also turned into the airways for the same purpose.

Where dust zones occur in a metal mine, as at loading chutes, transfers, and underground crushers, bag filtering units are installed to reduce the dust content of the air; in some cases spray chambers are installed and the air is forced through them for the same purpose.

ILLUMINATION

Stationary Lighting.—Underground lighting in metal mines is effected by electric lights where electric power is available. The principal points where lights are placed are at stations and along main drifts and crosscuts. The incandescent light of moderate candle power is very generally used. From two to four lights are placed at stations, depending on the size. In shafts where ladderways are used, which is often the case in shallow mines, lights are placed above each landing. In drifts lights are spaced at intervals of 100 to 200 ft. in straight portions, and a single lamp is placed at every turn. The lighting facilities should be exceptionally good wherever machinery is employed underground. Adequate illumination is a factor in the prevention of accidents.

For night work in open pits arc lights supported on masts are often used for general illumination. The mast should be supported on skids in such a manner as to enable the light to be conveniently moved. Gasoline flare lamps and portable acetylene lamps of large size are used where electric power is not available.

In coal mines incandescent lights protected by heavy glass outer globes and metal cages are used for general illumination at shaft bottoms, fresh-air inlets, switches, doors, and to some extent in rooms; semiportable lamps supplied by flexible cords are used in repair barns, for inspecting machinery, and to some extent in rooms and entries; projector types of lamps are used for floodlighting nongassy working places and areas; explosion-proof headlights are used on locomotives. All other types of lights are portable or have storage-battery sources of power. In gaseous mines permissible lamps are essential.

In shaft sinking in metal-mine operations, the shaft bottom is illuminated by a cluster of incandescent lamps with reflector and wire cage and a flexible cable unwound from a reel at the surface or other point from which sinking starts. Where power is not available, miner's lamps are used.

Reflecting signs for various purposes are made from half-inch reflecting buttons and are used both in coal and metal mines at critical places such as exits, switches, derails, safety stations, direction points, safety equipment stations, and first-aid stations.

Miner's Lamps.—Candles and oil and carbide lamps are used for illumination in small operations, prospecting, and exploration. In large, systematic metal-mining operations and in coal mining the electric miner's lamp is widely used and can be considered standard. The dry-battery flash lamp is an important emergency light. Portable electric flood lamps, storage-battery and dry-battery types, are also used in underground lighting where power circuits are not available and where better lighting is required than can be furnished by the miner's lamp. Lamp design has kept pace with the miners' requirements, and the market supplies excellent types for normal and emergency use.

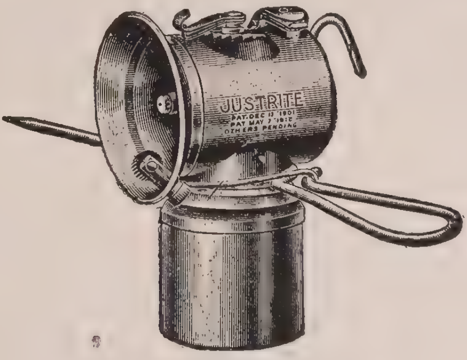

FIG. 137.—Miner's acetylene lamp.

The carbide cap lamp (Fig. 137) was widely used at one time. It carried enough carbide to burn about 4 hr., and an extra font and a supply of carbide more than sufficient for a shift's use was given the miner. The miner's canteen furnished the water supply. Hand carbide lamps containing sufficient carbide for a shift's use are also used. Carbide lamps give better illumination than candles and their cost is about half that of candle illumination. Where open lights are permissible in coal mining the carbide still continues to be used to some extent, but it is now largely displaced by the electric lamp.

The electric cap lamp was developed to give a better light than could be obtained by the oil, naphtha, or benzene safety lamp. A number of makes that meet the exacting standards of the U.S. Bureau of Mines are available. Where approved they are called permissible electric lamps. Such lamps may be used in gaseous atmospheres. They are sealed and so locked as to prevent tampering by the user. The lamp attached to the miner's cap is supplied with current by a flexible cord from a small storage battery attached to his belt. The battery carries a sufficient charge to meet the requirements of a shift's operation. The total weight of an electric lamp ranges from 3.6 to 5.3 lb.; usually two bulbs are provided in the cap piece, one for normal lighting and the other for emergency use. The storage battery is of the two- or three-cell alkaline or lead-acid type.

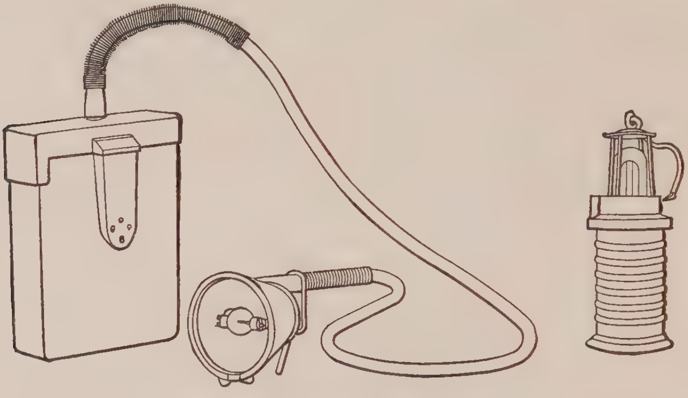

Fig. 138.—Electric mine lamps.

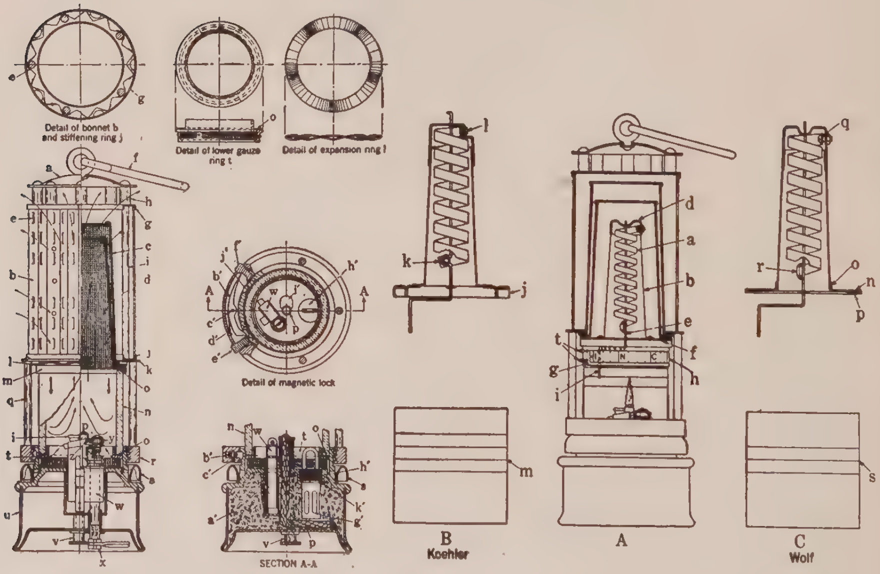
Detail of bonnet b and stiffening ring j
Detail of lower gauze ring t
Detail of expansion ring l
Detail of magnetic lock
SECTION A-A
B
Koehler
A
C
Wolf

Fig. 139.—Details of Wolf safety lamp on left—*a*, hood; *b*, bonnet; *d*, inner gauze; *e*, louvers; *f*, bail; *v*, wick adjuster; *w*, relighter. On right, improved Koehler (*B*) and Wolf (*C*) lamps with bimetal helix units and pointers to indicate methane content. *A*—Bimetal helix unit; *B* and *C* are mountings of unit. (*U.S. Bur. Mines.*)

Since 1928 the Bureau has recommended the use of the electric cap lamp of the permissible type in all coal mines. Metal mines experimented with the coal miner's electric cap lamp and found it to be so much more satisfactory than the carbide lamp that it is now in general use. See Fig. 138.

Flame-type safety lamps, Wolf, Koehler, and others, and permissible types with seal or magnetic locks and relighters preceded the electric lamps and are still in use to some extent. The latest permissible types of such lamps are now used for testing gaseous and oxygen-depleted atmospheres and are almost indispensable for this purpose. Figure 139 shows the construction of such a lamp and Table 60 gives weight data.

Electric lamps of the battery type have been designed for special purposes such as signaling, floodlighting, wear by animals, and hand use. For intermittent lighting and emergency use several kinds of permissible flashlights are available.

Lamp House and Servicing.—Systematic servicing and repair of miners' lamps are necessary, and at most mines a lamp room for storing and issuing lamps and a repair shop for maintenance and repair are provided. Battery-charging racks are required for electric lamps. Experienced lampmen are necessary where modern permissible lamps are in use, as the permissible flame safety lamp must be in prime condition before it can be used.

TABLE 60.—WEIGHT OF LAMPS, POUNDS (PERMISSIBLE TYPES)

Lamp	Approx. weight	Lamp	Approx. weight
Koehler, steel	3.37	M.S.A. hand lamp, model H (S.B.)	5.06
Koehler, aluminum	2.37	M.S.A. animal lamp, model K (S.B.)	6.31
Wolf, steel	3.31	M.S.A. floodlight, model E	18.53
Wolf, aluminum	2.84	Eveready flashlight	1.25
Edison, model J (S.B.)	5.1	Justrite special cap lamp (D.C.).	2.56
Wheat, model G.W. (S.B.)	5.2		
Signal—Ceag (S.B.)	8.06		
Justrite (D.C.)	11.00		

NOTE.—Koehler and Wolf lamps are of flat-wick design. S.B. = storage-battery type. D.C. = dry-cell type. M.S.A. = Mine Safety Appliances Company. Weights of Koehler and Wolf lamps are for empty lamps.

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CHAPTER X

SUPPORT OF MINE WORKINGS

PHYSICAL PROPERTIES OF ROCKS

Physical Constants.—The physical constants of the strength of rocks have not been determined consistently throughout the range of rocks encountered in mining. In addition the range of variation in any rock species is so great that wide generalizations are apt to be unsatisfactory. However, more or less information is available and the following table gives the average results for a number of common rocks.

Table 61.—Compressive Strength of Building Stones of Good Quality

	Lb. per cu. ft.	Comp. strength, lb. per sq. in.	Shearing strength, lb. per sq. in.	Depth at which comp. strength is reached
Granite	170	15,000	2,000	12,000 ft.
Sandstone	150	8,000	1,500	7,680 ft.
Limestone	170	6,000	1,000	5,082 ft.
Marble	170	10,000	1,400	8,470 ft.
Quartzite	176	10,000	. . . .	8,196 ft.
Slate	175	15,000	. . . .	12,340 ft.
Trap	185	20,000	. . . .	15,560 ft.
Anthracite coal	. . . . . .	1,500		
Bituminous coal	. . . . . .	1,100		

The tensile strength of rocks is low as compared with the compressive strength and is not accurately known except in a few cases. The modulus of rupture has, however, been determined in a number of cases and a few results are averaged and presented.

$$\text{Modulus of rupture} = R = \frac{3L}{2BD^2} W$$

where W = breaking load concentrated at center of test piece, L = unsupported length of rock prism, B = width of rock prism, D = depth of rock prism. The modulus of rupture is in excess of the tensile strength of a rock, while the shearing strength is generally slightly in excess of the modulus of rupture. Fine-grained rocks have usually a greater modulus of rupture than coarse-grained rocks of the same species and laminated rocks greater than holocrystalline, while holocrystalline rocks have moduli of rupture greater than those of sedimentaries. The magnitude of the modulus of rupture indicates the self-sustaining power of a rock where it spans an opening.

	Modulus of rupture, lb. per sq. in.
Slate	7,736
Granite	1,681
Sandstone	806

Most rocks have variable moduli of elasticity. Granites and crystalline rocks approach perfectly elastic substances, while sandstones and rocks of a similar nature take a permanent set for slight loads. For example, the modulus of elasticity for different granites varied from 5,128,000 to 8,160,000 lb. per sq. in., while for a single specimen of sandstone it ranged between 1,780,000 and 7,711,700 lb. per sq. in. (Merrill).

Coefficient of Expansion.—Merrill gives the following coefficients of expansion for a unit length of 1 in. and a temperature increment of 1°F.

Granite	0.000004825
Marble	0.000005668
Sandstone	0.000009532

Porosity of Rocks.—Porosity is of moderate importance in that very porous rocks act somewhat differently when saturated and subjected to loading as compared with the unsaturated condition. The pore space in per cent of volume is given:[1]

Granite	0.16– 1.2
Sandstone	10.22–15.89
Slate and shale	3.95
Limestone	4.85
Sand	48.00
Clay	45.00
Soils	55.00

Angle of Repose.—Loose rock when heaped forms an angle with the horizontal, termed the angle of repose. Values for different substances are given.

Clean sand	26.5°
Sand and clay	18.5°
Clay	16.0°
Clean gravel	26.5°
Soft rotten rock	45.0°
Broken hard rock	45.0°

The presence of water influences the angle of repose to a marked extent.

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[1] *Water Supply Paper, Bull.* 160, U.S. Geol. Survey, p. 61.

CLASSIFICATION OF ROCK MASSES

Rock masses can be most conveniently divided into five groups.

Group 1.—The rock mass is composed of coherent particles and shows no planes of weakness. It may be crystalline or composed of irregular grains cemented together. Most igneous and some metamorphic rocks fall into this group.

Group 2.—The rock mass shows a series of parallel planes of weakness such as the planes of stratification in a sedimentary rock or sheeting planes in an igneous rock. Such a rock mass is strongest in a direction transverse to the planes of weakness and weakest in a direction parallel with such planes. Sandstones, limestones, slate, shales, and sheeted igneous rocks are examples.

Group 3.—The rock mass shows two or more systems of planes of weakness. Any rock type may fall into this group, as planes of weakness are produced by dynamic and structural causes. Jointing, extensive fissuring, weathering, sheeting in several different planes, and even cleavage (the cleat in coal) will weaken an otherwise solid rock. A rock mass of this nature would be apt to cave. It is evident that the degree of weakening plays an important part in determining the strength of a rock mass falling within this group.

Group 4.—The rock mass is noncoherent. A rock mass of this type may be composed of coarse and fine fragments or entirely of finely divided material. Sand, fault breccias, loose rock, extensively fissured and broken rock, soils, etc., fall into this division.

Group 5.—The rock mass is semiplastic or plastic. Semifluid rocks, clay, hydrothermally altered rock in the presence of water, and clay shales are typical examples of such rocks. The content of water and the proportion of clay determine the degree of plasticity. Such rocks deform readily under pressure and tend to relieve the pressure by squeezing into openings.

It should be noted that a given rock mass may embrace one or more of the above groups. For example, the portion of an igneous rock mass near the surface may be so thoroughly weathered as to be noncoherent and properly classed in group 4. Deeper down it may grade into group 3, and at greater depth it may pass through group 2 and then into group 1 without sharp divisions between. In other words, a given rock mass may not be consistently within one group, but different portions may fall into different groups although the rock species be the same throughout.

The miner makes use of the terms "solid ground," "heavy ground," "loose ground," and "swelling ground" to indicate the degree of danger and the difficulty of support. Solid ground characterizes rock masses of group 1; heavy ground, groups 2 and 3; loose ground, group 4; and swelling ground, group 5.

A knowledge of rock types and their physical properties is an aid to the miner in determining the necessity for support in mine workings. Close observation of the prevalence of planes of weakness, the presence of watercourses, fissures, etc., in conjunction with sounding by a hammer and testing with a bar and pick, are important supplementary methods for determining the stability of a given rock mass.

Equilibrium of Rock Masses Undisturbed by Mine Workings.—Every rock mass is under more or less stress. This stress is due to gravity, changes of temperature, chemical changes such as kaolinization and hydration, atmospheric pressure, and gas pressures due to the liberation of gases within the mass. Static equilibrium is established by the balancing of all the external and internal forces acting upon the mass.

Gravitational stress is almost always the important force to be considered. Deeply buried rocks are compressed by the weight of the superincumbent mass. They are in much the condition of a piece of rock in a testing machine. In some mine workings "explosive rock" is encountered at great depth. This is nothing more than rock which has been compressed almost to the limit of its strength and is afforded an opportunity to expand into the working. The result is the flying of fragments accompanied by sharp reports, an action closely paralleled in the testing machine. Pillars that have begun to fail frequently show the same phenomenon.

Rock masses that have been exposed from deeply buried positions by erosion are often in a compressed state and when removed from position expand. This has been observed in some granite quarries.

The surface debris covering rock masses in positions where the surface topography is accentuated is in a condition of more or less unstable equilibrium. The creep of such material downhill has been observed by geologists, and this condition necessitates the careful supporting of mine workings where they penetrate formations of such a nature. Almost every miner has observed the marked tendency of tunnel entrances to cave and the readiness of the support placed in such positions to fail.

Equilibrium of Rock Masses Disturbed by Mine Workings.—When rock masses are penetrated by mine workings, a condition of unstable equilibrium is set up. The tendency is for the contiguous rock to fill in the working. This it may do by breaking off in small pieces and gradually filling in, the filling-in process requiring a long time, or it may close in quite rapidly where the rock mass is either plastic or greatly weakened. The physical nature of the rock mass, as well as the magnitude of the forces tending to restore equilibrium, determines the rapidity of readjustment. The general relation of the time element to the physical nature of the rock is shown in Fig. 140. The rate of readjustment for rocks of group 1 is practically zero; for group 2 it is very small; for group 3 of comparatively small magnitude but greater than 2; group 4 very great, and for group 5 it depends on the degree of plasticity, being very rapid for the more plastic rock masses.

Excepting rocks of group 5, all rock masses superimposed upon a mine working, whatever its magnitude, tend to fail in a similar manner. If we assume an unsupported working, failure first begins at the center by bending. In a weak rock mass this is quickly followed by caving. Failure of a strong rock may be indefinitely postponed; failure of a weak rock supported by timbering may be postponed for a considerable time. When failure has commenced it proceeds upward, the area of rock affected diminishing at successive altitudes until equilibrium is restored either by the supporting tendency of the archlike contour of the walls surrounding the zone of failure or by the filling of the working and the space immediately above by the caved material.

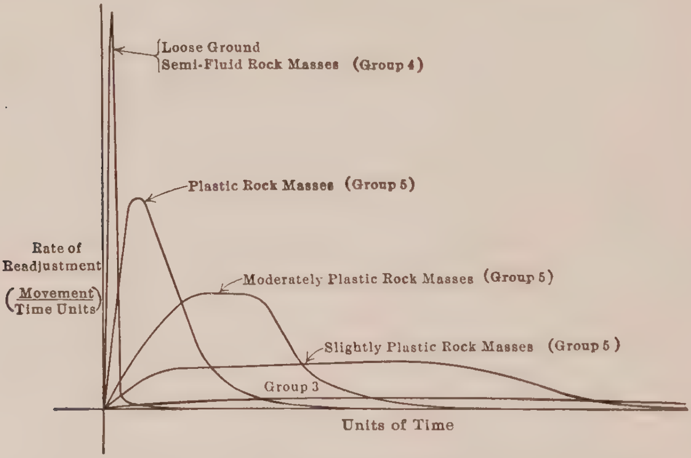

FIG. 140.—Rate of readjustment of rock masses in relation to time.

Where the roof rock breaks in pieces the height of the disturbed mass is comparatively limited, since the broken rock will occupy a greater volume than when in a solid condition. The increase in volume, about 50 per cent, tends to fill the space and bring about a condition of equilibrium. All rocks do not fail in this manner, and thus the height height of the zone of disturbance varies.

For convenience in the discussion the following terms are defined.

Zone of disturbance is the entire volume of rock above the working that is more or less broken and moved in the restoration of equilibrium.

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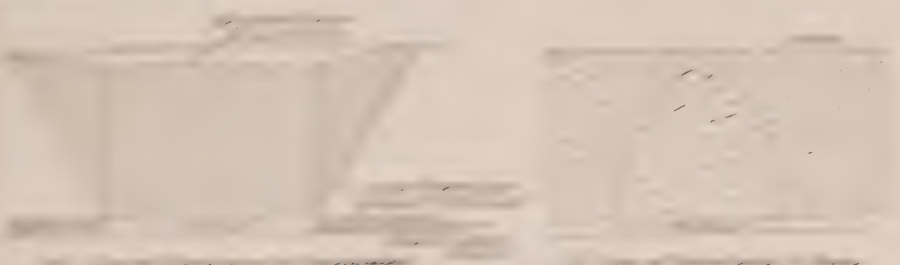

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librium is shown in the figure. In *b* a dome approximating a Gothic arch in section represents the zone of disturbance in a hard rock. These may be taken as the extreme types where the beds are horizontal.

Where beds and the working also are inclined, the zone is not symmetrical and its axis is inclined. The axis is more and more inclined, until it approaches the horizontal in proportion as the beds and working are inclined. The axis is inclined to the vertical and also to the normal to the strata. Where the zone of disturbance intersects several groups of beds at varying inclinations the axis of the zone is deflected in passing from one group to another and approaches the normal of the group in which it is. Faults and fissures or other interruptions to the continuity of the beds also influence the inclination of the axis.

Area and Height of Working.—The height of the zone of disturbance does not extend after a certain point has been reached, whatever the depth of the working. The height of the zone of disturbance will depend upon the amount of expansion of the material under movement and the volume of the space which it must fill in order to restore equilibrium. The following example will illustrate the point.

COAL SEAM 13 FT. IN THICKNESS—HEIGHT OF WORKING 13 FT.

	Maximum Height of Zone of Disturbance, Ft.
Area of working unlimited, without stowing	2,600
Area of working unlimited with stowing admitting of 40 per cent compression	1,050
Area limited to 164 ft. in breadth, without stowing	657
Area limited to 164 ft. in breadth, with stowing	327

In general where the area of the excavation is great the height of the zone of disturbance does not exceed 200 times the height of the excavation, but where the area is limited it is included between 2 to 4 times the breadth of the excavation.

Surface Subsidence.—The depth of the working below the surface influences this factor to a considerable extent. For very deep excavations equilibrium may be reached before the zone of disturbance reaches the surface, while for shallow workings the surface may intersect the zone of disturbance and a surface subsidence may result. The surface subsidence usually takes the form of a basin. The area of the surface affected may be smaller or greater than the area of the working, depending on the relation between the surface and the zone of disturbance and the nature of the rock mass. The area is less in hard rocks and usually greater in material like sand. In soft and compressible rocks the surface area coincides with the area of the working chamber for workings of moderate depth. The prevention of surface subsidence can be effected by leaving pillars, and it can also be materially reduced by stowing or filling.

Block Movement.—Underground workings in strong rock masses of relatively small extent, as is so often the case in metal mining, are bridged across by the rock mass and involve no special problem in support; but where the workings are extensive and large areas of the rock mass are unsupported, a general failure of the hanging wall may result. This is of the nature of faulting; the whole mass moves along a fault plane established by the line of failure of the mass. In the gold mines of South Africa and the Michigan copper mining district block movement has occurred and has brought to the attention of mining engineers the importance of studying the problem of mine support in some of its larger phases. In hard rocks fracture lines tend to prolong themselves; when once started they may extend to great distances. In an inclined working of low height and great area, the mass of the hanging wall may be likened to a cantilever beam. If the working intersects a plane of weakness, a wedge-shaped mass is released and the block becomes active. Almost every rock mass contains lines of weakness, and failure (movement) is inevitable where excessive stress is thrown upon it.

The usual method taken to prevent such a contingency is to leave pillars, which suffice where they are of sufficient areal extent and the rock is of sufficient strength. Where pillars are insufficient, filling is the only way to prevent general movement of this nature. Whenever block movement begins it is best to stop working until equilibrium has been restored by the closure of the openings.

Plastic Rocks.—Plastic rocks are active to a greater or less extent under the influence of the pressure caused either by their own weight or by that of the superimposed strata. They deform readily and as a consequence squeeze into workings quickly. The zone of disturbance is coextensive with the area of the working, and there is little or no tendency for them to expand. Top, lateral, and bottom pressures are developed in plastic rock masses. The plasticity of the mass determines how closely these pressures correspond with that due to the "head" of rock producing the pressure.

Sand and Loose Rock.—The action of pulverulent material under ground pressures can be deduced from the studies made of the pressure of grain in bins. The top pressure coming upon any given working is not that due to the weight of a column of the areal dimensions of the working and extending to the vertical height of the sand, but is caused by a much smaller mass. The mass is dome-shaped and of a height from 2½ to 3 times the diameter irrespective of the height of the sand above the working. The ratio given is for material consisting of rounded grains that can slip freely over one another. With material of the nature of broken or loose rock, which tends to lock or key, particle with particle, or the protuberances of which prevent free movement under pressure, the above ratio must be materially less. Lateral pressure is met with in loose rock, but this, unlike that encountered in the case of plastic rock, is relatively small while bottom pressure is absent.

Miscellaneous Causes of Lack of Stability.—The drainage of clays by mine workings produces more or less shrinkage and may thus start movement in the superincumbent mass. This movement may be expected to continue until the clay has been consolidated and equilibrium restored.

The action of air on freshly exposed rock masses produces drying, shrinkage or expansion, and general weakening in some cases with the resultant movement of the mass contiguous to the working. Water sometimes acts in a similar manner. Expansion and contraction due to temperature changes produce unstable equilibrium. Rock masses contiguous to worked-out portions of a mine are often stressed and sometimes badly weakened by the slow readjustment which takes place. Thus in metal mines where pillars have been left between worked-out and filled stopes, the pillars, as soon as they are attacked, often develop into ground which begins to move and must be carefully watched and handled.

SUPPORT

Supporting structures are used in mining operations to keep the workings safe and open for operation. In order to do this they must overcome the unstable equilibrium of the rock mass enclosing them. At most, mine workings have to be kept open for a comparatively limited length of time, and thus the function of the supporting structures is to delay the readjustment of the rock mass and to prevent it from closing in during this period rather than to afford a permanent support. Rock masses that adjust themselves rapidly require the prompt placing of supporting structures, and these must be either of sufficient strength to resist the pressures caused by the crowding in of the walls or of such a construction as to yield sufficiently to prevent failure. If they fail before the working is ready to be abandoned they must be replaced.

Rocks of group 1 require no support; rocks of group 2, comparatively light members in the supporting structure; rocks of group 3, medium-sized members; rocks of group 4, comparatively heavy members; and rocks of group 5, either excessively heavy members or a structure that yields as the presure builds up.

The walls and particularly the roof of a working tend to fail by the flaking off of small pieces or by the sudden caving of large masses. The supporting structure must prevent small pieces from falling and must support any large masses that may become loosened. Lagging serves the double purpose of preventing the small pieces from falling and of distributing over a larger area the resistance of the main members of the supporting structure.

In any given working, the pressure coming upon the supporting structure is characterized as top, side, and bottom pressure. The intensity of these pressures is a function of time and the nature of the rock mass. As has been mentioned previously, all three types of pressure characterize rock masses of group 5, top and side pressure groups 3 and 4, and top pressure alone group 2.

Mine workings comprise tunnels, drifts, crosscuts, shafts, shaft stations, pump chambers, stopes, and rooms. Mining practice has evolved a wide variety of more or less conventional structures for the support of the different types of workings, which will be discussed in the latter part of this chapter.

STRUCTURAL ELEMENTS

As might well be inferred from the preceding discussion, the principal element made use of in mine support is a compression member. The prop or post is the simplest example. Beams are used, but it is seldom that they are used alone to support a distributed load. Rather do they serve the double purpose of a beam and a strut. Thus a stull serves to resist the side pressure of the walls of a stope and also carries the floor on which the broken rock falls and is supported. In a shaft the wall plates act partly as beams and partly as struts. The dividers act as struts. In bulkheads the members of the structure act as beams. In masonry structures the arch is often used.

In ordinary structures the designer can proportion the members to meet definite loads that can in the majority of cases be quite accurately determined, but the mining engineer has to deal with uncertain loads that rarely admit of even approximate determination. He is thus put to the necessity of trial or of following the empirical rules established rather vaguely by mining practice.

Where approximate determination of the ground pressures is possible either by observation on structures in place or by a careful study of the nature of the rock masses, proportioning of structural members can be resorted to. A few examples are given below.

Computation of Proportions of Pillars.—Douglas Bunting has proposed a method for the calculation of the proportions of chamber pillars in deep anthracite mines where the dip of the seam is at a low angle.[1] His method calls for the assumption that the load on a pillar is equal to the weight of a prism extending from the top of the coal seam to the surface and of a width equal to the distance from center to center of the chambers. He first derives the relation between the crushing strength of a prism of coal and a cube of coal. It is as follows:

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[1] *Trans. A.I.M.E.*, vol. 42, p. 236.

$$\frac{\text{Strength of prism}}{\text{Strength of cube}} = 0.70 + 0.30\,\frac{b}{h}$$

h = height of prism.
b = least lateral dimension of prism.

He next derives the formula

$$yx = 1{,}000\left(0.70 + 0.30\,\frac{b}{h}\right)b$$

y = depth below the surface.
x = distance center to center of chambers.
b = width of pillar.
h = height of seam.

A crushing strength of 1,000 lb. per sq. in. is used in deriving the above formula. From this formula Bunting has calculated a series of tables, a portion of one being given in Table 62.

TABLE 62.—DEPTH BELOW SURFACE FOR VARIOUS CHAMBER CENTERS AND THICKNESSES OF SEAMS; WIDTH OF CHAMBER 24 FT.

Distance of chambers, c. to c., ft.	Width of pillar, ft.	Thickness of seam, ft.			
		4	6	8	10
40	16	760	600	520	472
45	21	1,062	817	694	621
50	26	1,378	1,040	871	770
55	31	1,705	1,268	1,050	919
60	36	2,040	1,500	1,230	1,068
65	41	2,381	1,735	1,411	1,217
70	46	2,727	1,971	1,596	1,367
75	51	3,077	2,210	1,777	1,516
80	56	3,430	2,450	1,960	1,666
85	61	3,786	2,691	2,144	1,816
90	66	4,143	2,933	2,328	1,965
95	71	4,503	3,176	2,513	2,115
100	76	4,864	3,420	2,698	2,265

In the calculations no allowance for arching action is taken into consideration, and it is therefore evident that the proportions given have an additional margin of safety over that obtained by taking a crushing strength of 1,000 lb. per sq. in.

Computation of Thickness of Cover.—The same author has computed the thickness of rock cover where coal mining is carried out beneath heavy wash.[1] His method is to consider the rock cover as a slab supported by the pillars of coal. He computes the thickness of the slab required, assuming a value of 100 lb. per sq. in. as a safe modulus of rupture for sandstone. His base formula is

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[1] *Trans. A.I.M.E.*, vol. 51, p. 177.

$$W = \frac{2bd^2}{L} R$$

W = distributed load plus weight of beam, in pounds.

$$\left.\begin{aligned} b &= \text{breadth of beam} \\ d &= \text{depth of beam} \\ L &= \text{span of beam} \end{aligned}\right\} \text{in inches.}$$

By taking 120 lb. as the weight per cubic foot he derives the approximate formula for a chamber width of 24 ft:

$$d = 1.50 \sqrt{d^1}$$

d = depth in feet of rock cover.
d^1 = depth to top of seam.

This equation would give the minimum thickness of cover where all conditions are absolutely known. Where conditions are not completely known, Bunting advocates increasing the result obtained by the above formula by quantities ranging from 5 to 50 ft.

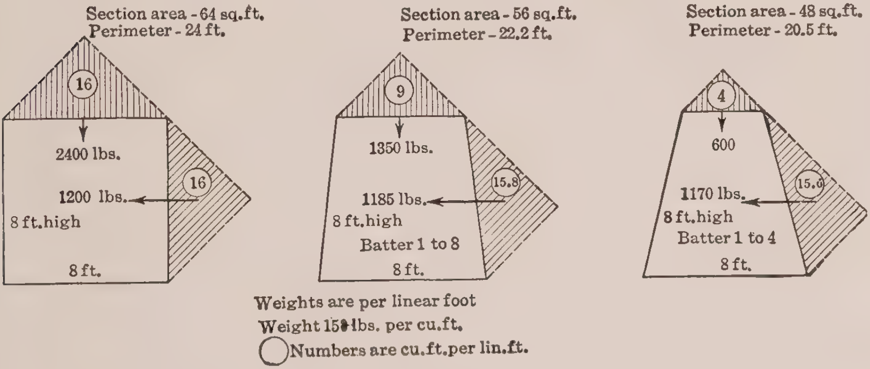
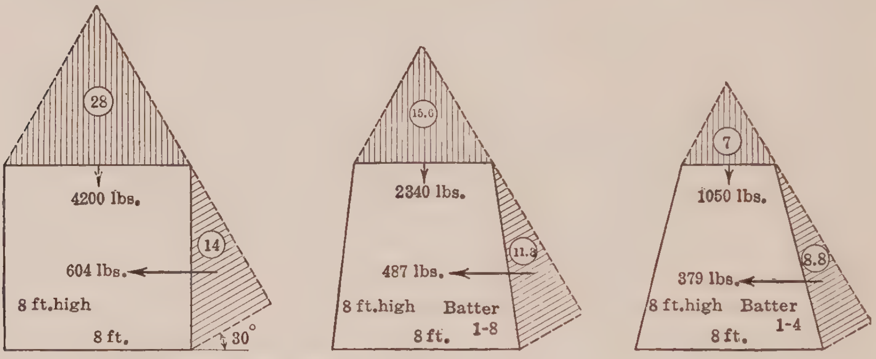


Fig. 142.—Forces acting upon drift timbers. Assumed angle of repose, 45 deg.

Support of Tunnels and Drifts.—Let us assume an angle of repose of 45 deg. in loose ground. Let it also be assumed that the loose rock above a plane of 45 deg. be supported by reaction within itself. The vertical and side loads on a timber support can then be calculated. They are graphically shown for an 8- by 8-ft. opening in Fig. 142. The effect of inclining the side posts is shown also. In calculating side pressure a coefficient of friction equal to 0.5 was taken. The loads are given in pounds per linear foot of drift. If we assume that the loose rock above a plane of 60 deg. to the horizontal is in equilibrium and the angle of repose is 30 deg., then the top and side loads will be as represented in Fig. 143. The shaded portions in all the figures represent the rock masses that must be kept in equilibrium by the reaction of the timbers or other supporting members. The two assumed cases are no doubt extreme, and somewhere between them lie the true top and side loads. It may be objected that there is not sufficient experimental proof of the above theory. This is true as far as any experimental work under mining conditions is concerned, but we have abundant proof from the study of pressure in grain bins that the theory made use of in the above calculations is not greatly in error. For example, various experimenters have noted that the bottom pressure in a grain bin remains practically constant when the depth of grain reaches a height equal to from 2 to 2½ times the width of the bin. Grain consists of more or less rounded particles that are free to move past one another and to adjust themselves to pressure. In loose rock we have a number of rough particles which key one with the other and which do not possess the same mobility that grain does. It is reasonable to suppose that the equilibrium plane will be flatter than that found experimentally for grain. The term "loose rock" is not a definite one and is applied to all degrees of looseness ranging from fractured rock, which is so thoroughly keyed as to be almost self-sustaining, to rock masses that would move on the least disturbance.



Weights are per linear foot
Weight is 150 lbs. per cu. ft.
○ Numbers are cu. ft. per lin. ft.

Fig. 143.—Forces acting upon drift timbers. Assumed angle of repose, 30 deg.

If the timber structure is calculated for the extreme condition of looseness of the rock mass and a reasonable factor of safety is used, the resulting structure should meet the requirements. Experience and judgment will indicate how great a departure must be made from the dimensions found by calculation.

For plastic rocks, the degree of plasticity varying between wide limits, it is evident that the above method of calculation is inapplicable. With the plastic rocks there is a certain approximation to fluid pressure, and as a consequence the loads coming upon the support are often greater in magnitude than those produced by loose rock masses. There is no method, other than the one of trying supports of known strength, for the determination of the top, side, and bottom loads produced by plastic rocks. For material like sand it is debatable whether the top weight would not be greater than that shown in Figs. 142 and 143. Sand would act more nearly like grain, and in proportion as it approximated this action it would appear reasonable to suppose that a greater height above the cap should be taken than that given by the 60-deg. plane.

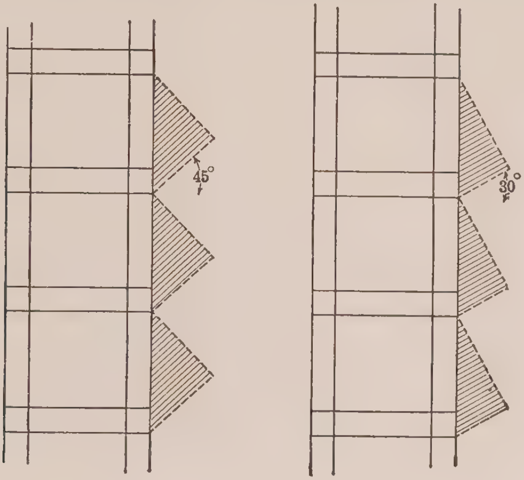


Fig. 144.—Forces acting upon shaft timbers.

If the above theory is applied to a shaft, the conditions for the two equilibrium planes, 45 deg. and 30 deg., would be as shown in Fig. 144. The tendency of the side pressure would be to force the sets down, and this action has been repeatedly observed in shaft-sinking operations in

Table 63.—Sandstone—Weight 150 Lb. per Cu. Ft.

Spacing of sets c. to c.	Span in feet. Weights in pounds							
	4	5	6	8	10	12	14	16
1	600	938	1,350	2,400	3,750	5,700	7,350	9,600
2	1,200	1,876	2,700	4,800	7,500	11,400	14,700	19,200
3	1,800	2,814	4,050	7,200	11,250	17,100	22,050	28,800
4	2,400	3,752	5,400	9,600	15,000	22,800	29,400	38,400
5	3,000	4,690	6,750	12,000	18,750	28,500	36,750	48,000
6	3,600	5,628	8,100	14,400	22,500	34,200	44,100	57,600

loose ground. The pull on hanging bolts becomes sometimes so great as to necessitate supplementing them with wire ropes.

In Table 63 the weights required to be supported by the cap piece have been calculated for different widths of opening and for spacings of the sets from 1 to 6 ft. An equilibrium plane of 45 deg. and a weight of 150 lb. per cu. ft. have been assumed. The corresponding timber dimensions (round yellow-pine timbers) for supporting these loads are given in Table 64.

TABLE 64.—CAP DIMENSIONS[a]

Spacing of sets c. to c. in feet	Span in feet. Numbers are diameters in inches							
	4	5	6	8	10	12	14	16
1	4	4	6	6	8	10	10	12
2	4	6	6	8	10	12	14	16
3	4	6	8	10	12	14	16	16
4	6	6	8	10	14	14	12″I (40)	15″I (42)
5	6	8	8	12	14	16	15″I (42)	2–12″I (31.5)
6	6	8	10	12	14	16	15″I (42)	2–12″I (40)

[a] The last three numbers in the two right-hand columns refer to steel beams. The weight per foot is given in the brackets. The factor of safety varies and approaches a minimum of 4.

MATERIALS USED IN SUPPORTING STRUCTURES

Timber, steel, brick, stone, concrete, and cast iron are used either alone or in combination for the construction of mine supports. Both hard and soft woods are used, but preference is given to the long-fiber softer woods. Hard woods are used where they alone are available or where some special reason indicates their use. The more important

TABLE 65.—PHYSICAL CONSTANTS OF SUPPORTING MATERIALS

	Lb. per cu. ft.	Compressive strength, lb. per sq. in.	Tensile strength, lb. per sq. in.	Shearing strength, lb. per sq. in.	Compression across grain
Hemlock	29.4	5,300	8,700	2,650	1,100
Chestnut	41.0	5,300	10,500	1,500	1,600
White pine	25.6	5,400	7,500	1,500	1,000
White oak	46.35	7,000	14,800	2,480	1,600
Shortleaf pine	43.6	8,500	15,900	4,425	1,200
Oregon pine	32.14	6,500	12,000	4,700	1,000
Redwood	26.23	3,500	8,500	....	800
Stone	165.0	6,000	....	1,500	
Brick	125.0	3,000	275	1,000	
Concrete 1–2–4	150.0	2,400	125	400	
Cast iron	450.0	90,000	25,000	18,000	
Medium steel	490.0	60,000	60,000	50,000	

physical properties of the above-mentioned materials are given in Table 65.

In addition to the physical properties, the cost and the life of the material must be considered. An approximate comparison of these factors, as well as of the physical properties, is given in Table 66.

TABLE 66

	Comp. compressive strength	Comp. tensile strength	Comp. shearing strength	Cost per cu. ft.	Cost, equal compressive resistance	Life in years
Timber........	1.0	1.0	1.0	$0.36	$0.36	3
Stone..........	1.2	....	0.37	0.20	0.166	Indef.
Brick..........	0.6	0.02	0.25	0.32	0.53	Indef.
Concrete.......	0.5	0.01	0.10	0.30	0.60	Indef.
Cast iron......	16.4	2.0	4.5	9.00	0.54	Indef.
Medium steel...	10.93	5.0	12.5	12.25	1.12	15–20

Timber is perhaps the cheapest material that can be used in the majority of cases. While it is satisfactory from a structural standpoint, its comparatively short life is an objectionable feature. The life of mine timbers is variable and depends upon a number of conditions. Dry, seasoned timbers last longer than green timber; peeled green timber longer than unpeeled; timber kept constantly wet in a good current of air will last indefinitely; timber in a hot, moist atmosphere will rot very rapidly; timber alternately wet and dry will rapidly deteriorate; timber subject to the attack of insects, as in the tropics, has a short life. The average life of a prop in Illinois coil mines varies from 1 to 4 years. The U.S. Forestry Service estimates the average life of an untreated mine prop in Eastern coal mines as 3 years.

Timber treatment has made marked progress in coal- and metal-mining practice. The principal preservatives in use are coal-tar creosote, preservatives derived from coal-tar creosote, zinc chloride, and, to some extent, sodium fluoride. Arsenic compounds, mercuric chloride, copper sulphate, and various commercial preparations have also been tried out. Ac-Zol, for example, is a mixture of copper, zinc salts, and phenol held in solution by ammonia. Wolman salts consist principally of sodium fluoride along with other toxic agents. Preservatives are applied by submerging the timbers in open tanks containing the solutions, by painting or spraying, and by the use of pressure chambers. In the last-named method the timber may be subjected to vacuum and then pressure in contact with the solutions or simply to pressure in contact with the solutions. The most effective treatment is by vacuum and pressure. The primary objective is to secure a deep penetration of the wood by the preservative. Dipping, soaking, and painting or spraying secure only a superficial penetration and are less effective. Metal-mining companies have made use of zinc chloride solutions and both open-tank and pressure-tank impregnation. Sodium fluoride is effective but generally too costly. Creosote treatment is also effective but not widely used, although it is sometimes cheaper than most other preservatives. For large timbers, pressure-tank treatment with coal-tar creosote costs from 24 to 34 cents per cu. ft.; pressure-tank treatment with solutions of ½ lb. per cu. ft. of zinc chloride costs from 15 to 22 cents per cu. ft. and of 1 lb. per cu. ft., from 18 to 24 cents per cu. ft.; smaller timbers cost more, as the absorption and handling are relatively greater. At the Miami copper mine the cost for treating timbers in 1930 was $6.32 per 1,000 bd. ft., the retention of zinc chloride being 0.7 lb. per cu. ft. Treated timber is used in shafts, stations, adits, airways, for track ties, and wherever long service is expected. In stoping operations treated timber has no economic advantage.

From a structural standpoint steel is an ideal material. It has the additional advantage of reducing the size of the excavation for given clearance dimensions. Its cost and the greater amount of difficulty and skill required in its use are objections. It is also subject to corrosion, and where used in the presence of water, particularly acid water, it must be painted with suitable protective paints.

Where permanency is the most important consideration, masonry of brick, concrete, or reinforced concrete is used. On the whole, masonry involves a much greater cost than timber, but this is warranted where long life is required. Reinforced concrete is used more than other forms of masonry for mine support, and particularly where the structural members must meet more or less tensional stress. For arches and circular lining, reinforcement is used only where ground pressures are severe.

Economic Principles.—Selection of materials and proportioning of members are predicated on safety, service, and cost. The engineer must choose between low initial and high maintenance costs and high initial and low maintenance costs. Frequently he is compelled to use what is at hand. The important requirement is safety. This involves a time element: The supporting structure should be safe until the working has been finished and the ore taken out and there is no longer any need of maintaining it. For a shaft the time may range from 10 to 30 years, for a drift from 2 to 5, and for an entry from 1 to 3 years.

Knowledge of the size of the ore body, the rate of working, the life of the mine, and the life of different parts of a mine is essential for the proper selection of materials. Too often the practical miner is allowed to have his say, and as a consequence unnecessary expense is required for maintenance. The practical miner neglects the time element. What will answer the purpose for the moment is often the only condition met.

The mining engineer should consider each part of the mine as a separate problem, and he should determine in each case: The purpose of the opening, the size and other dimensions, the length of time it is to be kept open, the nature of the ground, the materials obtainable, and the comparative costs with different materials for the time period involved.

Comparative Strength of Mine Timbers.—The comparative strength of timber beams, struts, I beams, and H beams can readily be determined by applying the principles of the strength of materials. For purposes of illustration two tables are presented. Table 67 gives the comparative safe loads for round yellow-pine beams and steel H sections in units of 1,000 lb., and Table 68 the safe loads for timber and steel struts. It should be noted that the "safe loads" are those that would be used in building construction. Under mining conditions it is an open question whether larger loads would not be permissible.

TABLE 67.—COMPARATIVE STRENGTH OF ROUND YELLOW-PINE AND STEEL H STRUTS, 1,000-LB. UNITS

Diameter, in.	Height, ft.				
	6	8	10	12	14
4	8.5	7.14	5.78		
6	22.17	20.13	18.1	16.06	14.02
8	41.2	39.41	36.7	33.98	31.27
10	64.4	64.4	61.58	58.18	54.79
12	....	....	92.74	88.67	84.60
14	....	....	....	125.44	120.69
16	....	....	....	....	163.06
4H	44.42	38.96	33.5		
5H	65.66	59.68	53.7	47.71	
6H	....	81.1	74.77	68.44	62.11
8H	....	....	119.37	112.32	105.28

NOTE.—Safe loads are based on N.Y. building law. For yellow-pine compressive unit stress, $1{,}000 - 18\frac{l}{d}$, where l is unsupported length and d is least diameter.

Carnegie Steel Co. steel mine timbers.

Preparation of Mine Timbers.—Where timber is used in irregular lengths and sizes it is frequently cut to dimensions and framed at the face. This is not an economical practice. In well-managed mines timbermen are required to measure the necessary dimensions and send their orders for timbers to the timber-framing department, where the timbers can be more economically cut and prepared. Where systematic timbering is a feature of the mining practice, special timber-cutting tools can be designed and installed. Standard machines for framing square-set timbers are available. Hand framing is necessitated in many cases, and this can be greatly facilitated by cutting a full set of templates and using them in laying out the work. The usual equipment of a timber-framing plant consists of a power saw of the swing type, a sawing bench equipped with rollers for handling long timbers, a small circular saw and bench for cutting wedges and miscellaneous use, and a timber-framing machine where square sets are used.

TABLE 68.—COMPARATIVE STRENGTH OF ROUND YELLOW-PINE AND STEEL H BEAMS, 1,000-LB. UNITS

Diameter, in.	Span, ft.					
	6	8	10	12	14	16
4	1.12	0.84	0.67	0.56		
6	3.77	2.83	2.26	1.89	1.62	
8	....	6.7	5.36	4.47	3.83	3.35
10	....	....	10.47	8.73	7.48	6.55
12	....	....	....	15.08	12.92	11.31
14	....	....	....	....	20.53	17.95
16	....	....	....	....	....	26.81
4H	9.84	7.38	5.9	4.92	4.22	3.69
5H	17.46	13.09	10.47	8.73	7.48	6.55
6H	27.59	20.69	16.55	13.8	11.82	10.35
8H	....	40.5	32.4	27.0	23.15	20.25

NOTE.—Safe loads for yellow pine based on a bending stress of 1,200 lb. per sq. in. and shearing stress of 70 lb. per sq. in. Loads are for seasoned timber only.

Safe loads for steel H sections based on extreme fiber stress of 16,000 lb. per sq. in.

Carnegie Steel Co. steel mine timbers.

TUNNELS, DRIFTS, CROSSCUTS, ETC.

The various conditions in different kinds of mines and mining districts have resulted in a great variety of designs for the support of tunnels, drift, crosscuts, etc. It is more convenient to classify designs by the materials employed, as, for example, timber construction, timber and steel, timber and masonry, steel and masonry, and steel, masonry, and reinforced concrete. In practically all cases the top pressure coming upon the structure is the most important, the side pressure is less and often needs no provision, and the bottom pressure the least, usually requiring no attention whatever. Support is unnecessary for firm rocks that suffer no appreciable disintegration, and very frequently the tunnel section is arched in order to make it still stronger. This is important where the section is large.

Timber Construction.—The simplest construction consists of a cap supported by vertical posts (Fig. 145*a*). This is used for top pressure alone. Round or board lagging is used. Where there is slight side pressure a wooden pin can be used, as shown in the accompanying figure. With square timbers and moderate side pressure the structure shown in *d* is used. Where top pressure is very great the cap is shortened by giving the posts a batter (*c*, *d*, *f*, *g*). The section of the set is trapezoidal, a type common in metal mines. Sills are used where an insecure foundation gives trouble. They are placed either across the working or longitudinally, and often a footboard is used instead of the sill. The distribution of the posts, caps, sills, lagging, and braces or sprags is shown in *g*. Where top, side, and bottom pressure must be resisted the structure shown in *b* or *e* is used. Timber sets are placed at a maximum distance apart of 6 ft., center to center. With a given size of timber closer spacing is used for greater pressures; if this does not meet the situation, larger timbers are required.

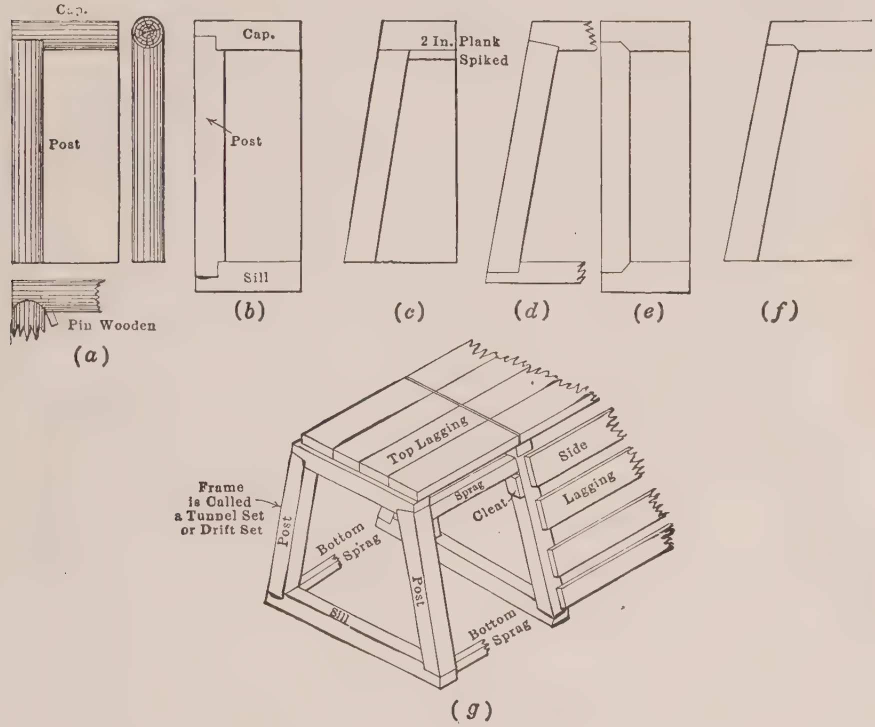


Fig. 145.—Drift sets.

The methods used in coal mines are shown in Fig. 146. The single-piece set consisting of a cap supported by hitches cut in the side walls is shown in *a*. In *b* the side walls are weak and posts are used. In *c* one post and cap is used. In *d* sufficient coal is left in the roof to prevent air from coming into contact with the roof formation. This frequently is all that is necessary. In *c* the short posts are foundationed in niches set on a hard coal layer. In *f* two lines of posts with wide headboards are used and answer the purpose where the roof is weak but does not flake off in small pieces. In *g* a method for supporting wide entries where only timber of moderate section can be obtained is illustrated. Coal-mine entries are from 8 to 12 ft. wide, and very heavy timbers would be required if the ordinary three-piece set were used.

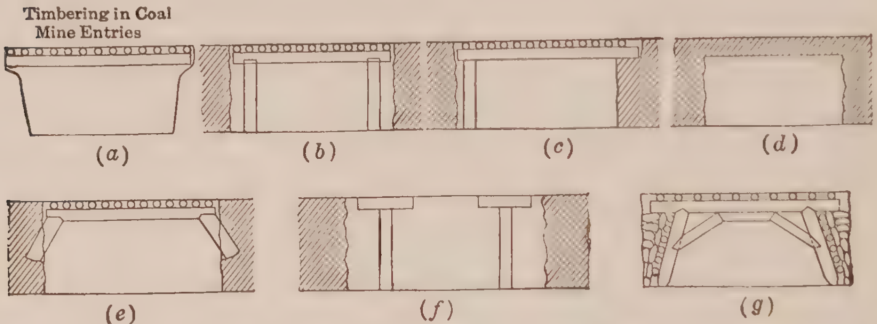


Fig. 146.—Timbering used in coal-mine entries.

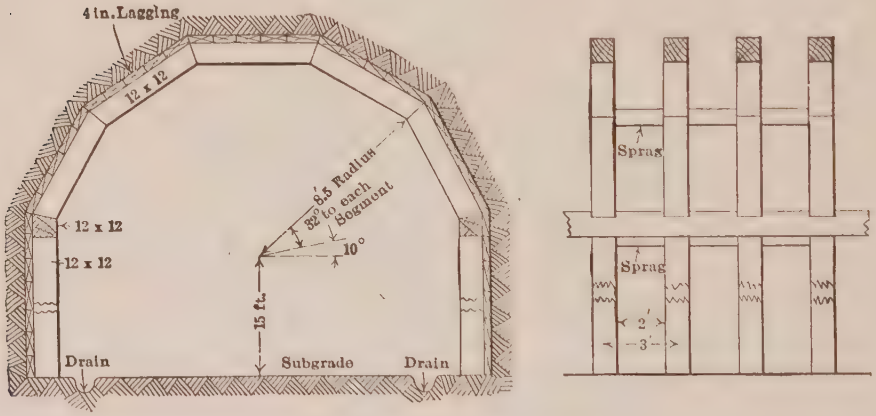


Fig. 147.—Railroad tunnel timbering.

Figure 147 shows a method of supporting a single-track railroad tunnel. This involves a segmental timber arch of five pieces and posts. This set modified in several ways is used for the support of pump chambers, stations, or very wide headings.

Timber and Steel.—One of the simplest methods of reinforcing timber caps is shown in Fig. 148*a*. In this method a piece of used steel cable is stapled on the underside of a cap and serves to lend some support. In order to be effective such a cable must be securely fastened.

In *b* are shown combinations of steel caps and timber posts. Used steel rails serve for caps and have the advantage that when they become bent they may be removed, straightened, and reused. *c* shows several designs for cast-iron chairs which are placed on top of the post and in which the steel cap is held.

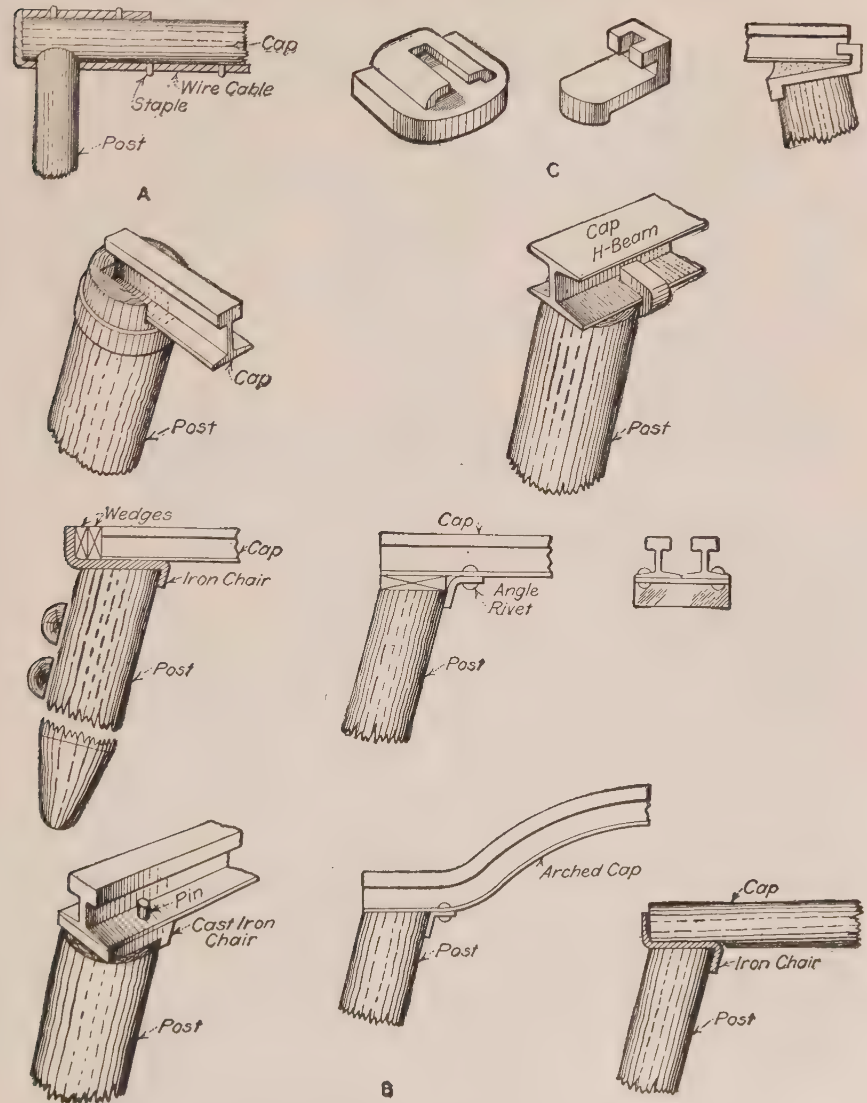


FIG. 148.—Steel and timber drift supports.

Timber and Masonry.—Dry walling and timber construction are used in the Lake Superior mines, and where suitable waste rock for the packs can be readily obtained it is a satisfactory and economical construction. Figure 149*a* shows the type of construction. The short round timbers

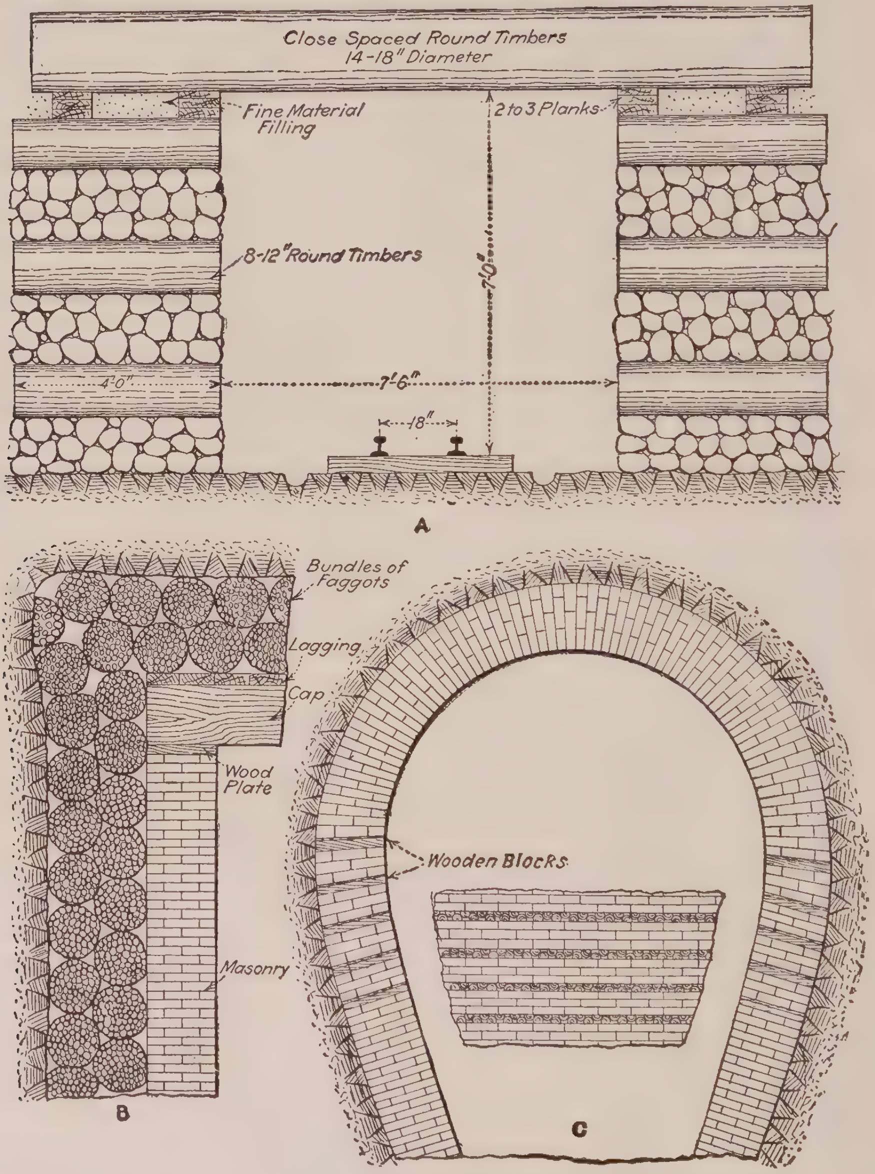


Fig. 149.—Timber and masonry supports.

in the side walls are used to bond the wall. In *b* is shown a type of construction used in swelling ground. The space between the timber and masonry structure and the walls of the excavation is filled with bundles

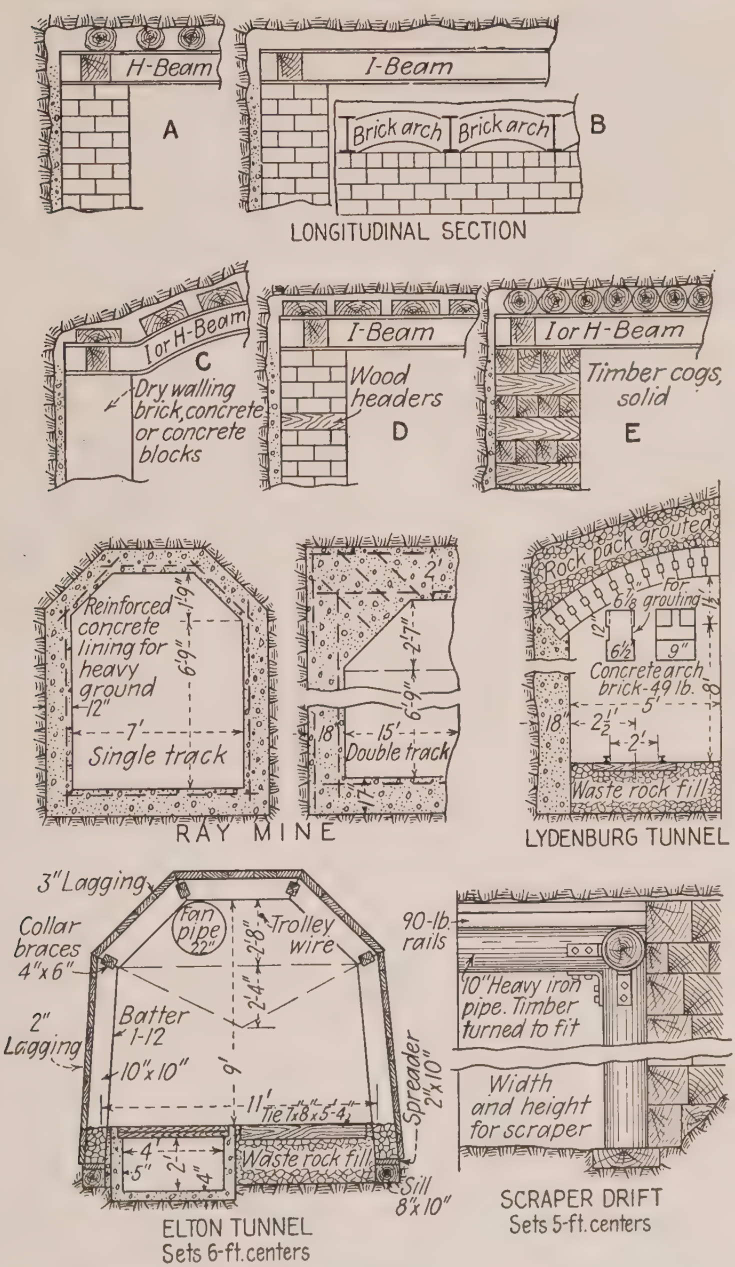


Fig. 150.—Examples of supporting structures for drifts, crosscuts, and tunnels.

of fagots. The condensation of the bundles serves to prolong the life of the structure.

In heavy ground where masonry structures are used, wooden pieces are built into the side walls in order to give elasticity to the structure. Fig. 149*c* shows the method. By using a composite structure of this nature the life of moderately heavy masonry structures can be greatly prolonged.

Steel and Masonry.—The simplest structure is a brick wall on either side of the drift and steel H beams or I beams across. The lagging rests on the steel members. This can be varied by building brick arches between the steel I beams. By arching the steel members as shown in Fig. 150*c* a more effective resistance to the top weight is obtained. Where more elasticity is desired, wooden members can be built into the brick side wall (*d*); and where still more "give" is required the side walls can be constructed of solid cribbing, as shown in *e*.

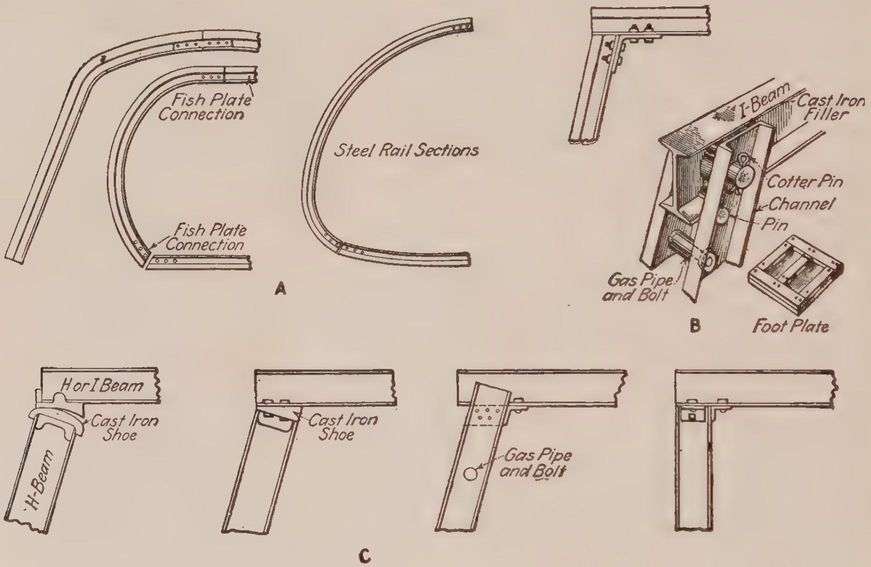


Fig. 151.—All-steel supports.

Steel.—Steel rails, H beams, I beams, and channels are used. Sets constructed entirely of steel are used to support lagging in permanent openings. They have the objection of moderate elasticity. Greater skill is required for erection, and they are difficult to cut where variable length is required for the "posts." The set can be designed along the lines of the timber set or may consist of curved or arched members. Figure 151*a* shows rail sets bolted together by fishplates. *B*, 1, shows a bolted joint used with steel rail cap and posts. *B*, 2, shows a set with I-beam cap and channel posts. A metal footplate is also shown. The

I beam is joined to the channel irons by two pins.[1] In *c* are shown various types of joints. Metal footplates are objectionable in that they are placed in a position where they are readily corroded. J. M. Humphrey advocates the use of wooden butts on the bottoms of the posts. He points out that the wooden butts add to the elasticity of the set, can be readily adjusted to length by cutting, and keep the bottoms of the posts from the floor of the drift.[2]

The structural shape best adapted for underground use is the H beam. It is manufactured in the sizes given in Table 69 and is equivalent in strength to certain sizes of round pine timbers that are also given in the table.

TABLE 69.—EQUIVALENT STRENGTH OF H BEAMS

4 in., 13.6 lb. per ft. = 8-in. round pine timber.
5 in., 18.7 lb. per ft. = 10-in. round pine timber.
6 in., 23.8 lb. per ft. = 12-in. round pine timber.
8 in., 34.6 lb. per ft. = 15-in. round pine timber.

Masonry.—Brick, dry walling, and concrete are the materials most used. The section is necessarily one in which there is an absence of tensile stress. For this reason semicircular, elliptical, oval, and circular sections are used. The inelastic nature of masonry requires that the structure be built strong enough to carry the loads, or else compressible material such as timber must be interposed between the outer wall of the masonry and the inner wall of the excavation. In European mines masonry is common in both coal and metal mines, but in America only a moderate use is made of these materials.

Reinforced Masonry.—Steel concrete is coming into considerable use and where permanency is desired is one of the most satisfactory materials. As with masonry, sections have to be designed that are reasonably free from tensile stress; they are of the same general nature as those used in masonry construction. Examples are shown in Fig. 150.

Wide Headings.—The wide headings required at colliery shaft bottoms necessitate either steel I beams and steel props or arching with reinforced concrete. An example of a 16-ft. entry is shown in Fig. 152. Steel reinforcement used with concrete arching is distributed as transverse rings placed within 2 or 3 in. of the inner wall, and longitudinal bars tied to the rings and spaced from 8 to 12 in. apart. Examples of the design of reinforced-concrete beams for wide headings are given in Fig. 153. The beams are reinforced with discarded 1.25-in. hoisting cables. The distribution of the reinforcement is shown in the figure. The load, 2,275 lb. per ft., is the maximum for the span and size of beam given. For smaller loads a given beam could be increased in length and used for a wider span.

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[1] Carnegie Steel Co.

[2] *Min. Press.*, Mar. 6, 1915, p. 372.

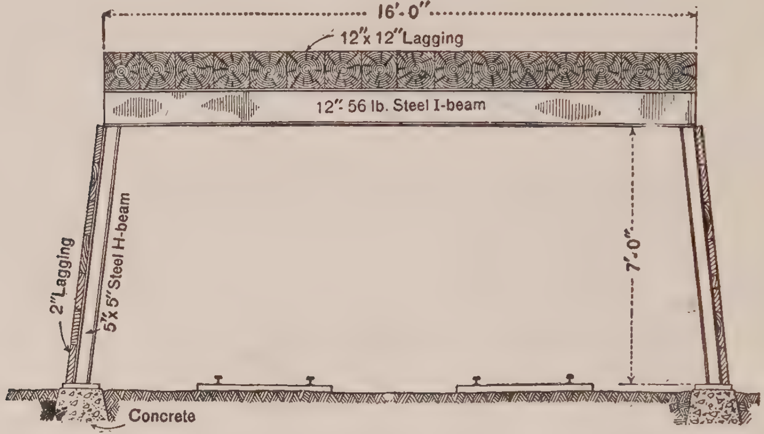


FIG. 152.—Steel set used for a main entry. (*Illinois Coal Mining Investigations.*)

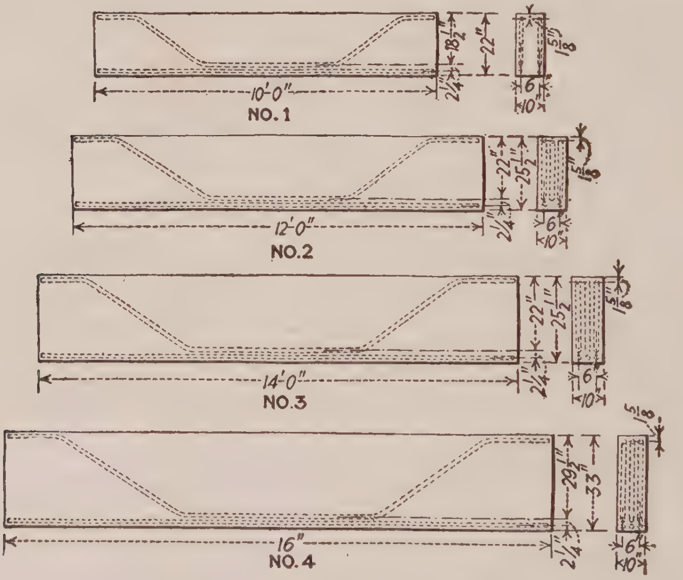
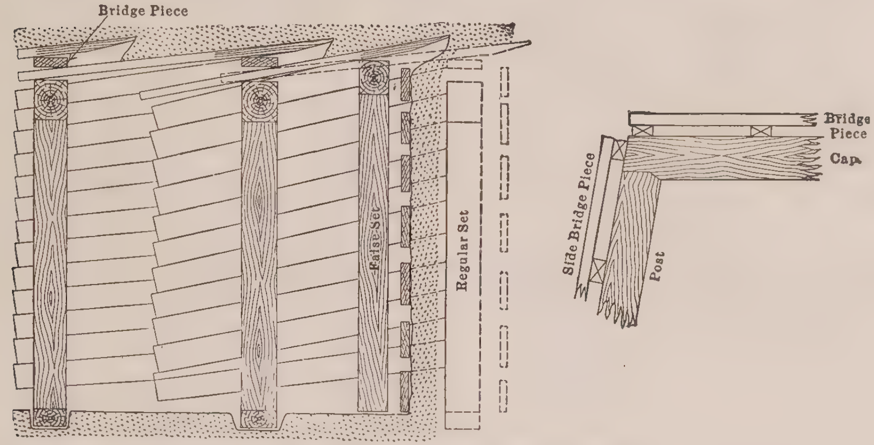


FIG. 153.—Reinforced-concrete caps. (*Eng. Min. Jour.*)

Timbering in Loose Ground.—The method commonly used is known as forepoling or spiling. Bridge pieces are placed on both cap and posts, and chisel-shaped lagging boards (2 by 6 or 3 by 6 in.) are driven within the space. The ground is picked away near the top of the section as the top boards are driven. When they are about half driven a false set is placed in position and the overhung ends of the boards are supported.

They are then driven to their final position, the face is squared up, and a set is placed in position. The false set is then removed and the face is ready for a fresh advance. Sideboards are angled upward and outward and are driven first at the top of the drift and in sequence to the bottom. In very loose ground the face must be supported by boards (face boards). Where face boards are required they must be placed twice for each advance of a regular set, once for the false set and once for the regular set. The face boards are wedged between the ends of the sideboards.



Forepoling in Soft Ground **Sprags between Sets not shown.**

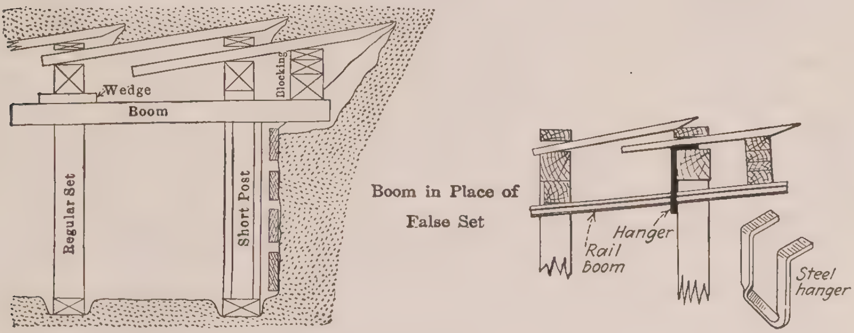


FIG. 154.—Timbering in loose ground.

In place of the false set a boom is often used. This is a long timber supported by a short post, one end of which is blocked up so as to support the ends of the top boards, the other being held down by blocks and wedges interposed between the end of the boom and the cap of the next to the last regular set. By its use the placing of face boards in only one position for each fresh advance is required. Figure 154 shows both methods. In place of the boom and post, steel hangers and steel rails can be used to support the boards, this method giving more room at the face. The arrangement is shown in the sketch, Fig. 154.

Intersections.—Intersections of crosscuts and drifts and main and side entries require a relatively wide opening to be supported. In Fig. 155 several different methods are shown. Such places can also be advantageously supported by the use of steel I beams.

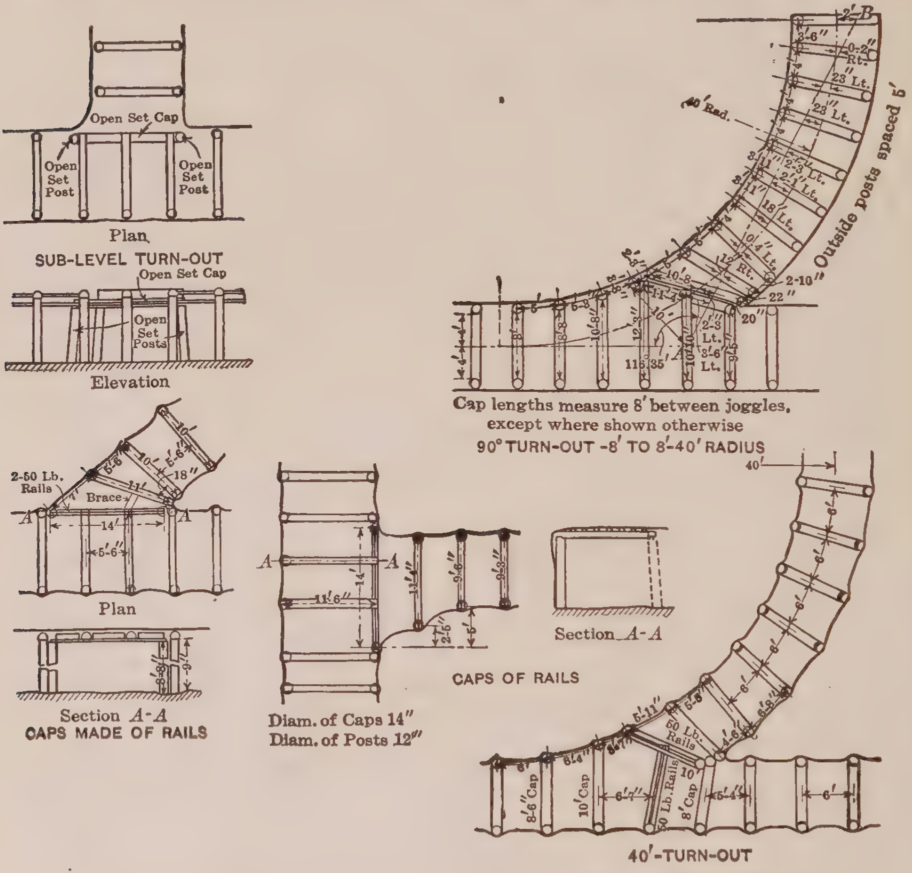


FIG. 155.—Timbering used in Mesabi mines at intersections. (*Eng. Min. Jour.*)

Stations.—Shaft stations are from 15 to 20 ft. in width and are supported by three-piece sets in firm ground, by three-piece sets and knee pieces in less firm ground, and by three- or five-piece wooden arches in heavy ground. Opposite the station, the wall plates of two- or three-shaft sets are omitted, and heavy vertical timbers used in place of the studdles. These timbers are set in hitches at top and bottom, or the reaction inward is taken care of by a line of braces extending from the front to the back of the station. Station sets are usually placed on 4-ft.

centers; in very heavy ground they may be placed on 2-ft. centers or set to set. The height of the station is designed to meet the special requirements entailed by the length of timbers and machinery required to be unloaded. The height varies from 10 to 16 ft. Figure 156 illustrates the method of timbering a station at a vertical shaft.

Pump Chambers.—Pump chambers are essentially the same as shaft stations. Where possible they should be constructed in solid rock requiring no support. In heavy ground the permanent nature of the structure warrants the use of steel or steel and concrete, although timber is used in many Western metal mines. Three-piece sets and three- or five-piece arched sets are used. Jacket sets are often used in very heavy ground.

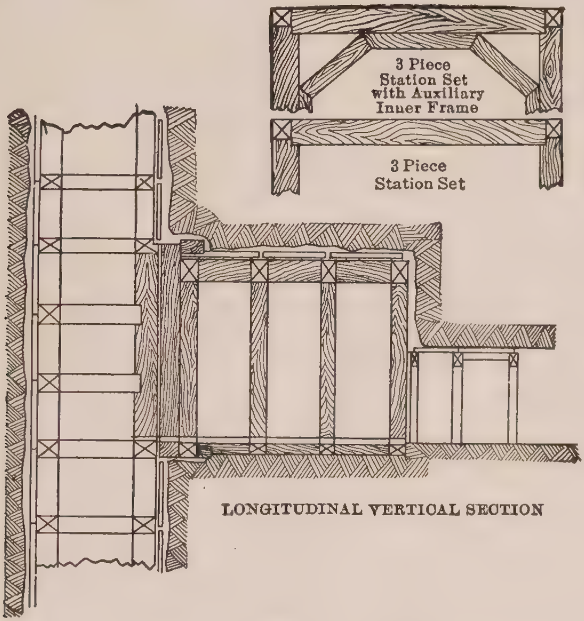


FIG. 156.—Station timbering.

SHAFTS

Shafts are the working openings of many mines, and as a consequence must be constructed to last the life of the mine. Where it is possible the shaft should be driven in a formation which is least likely to be disturbed by the mining operations and which is in itself solid and free from weakness. Usually the footwall is the best location, although conditions often require the shaft to be in the hanging wall or in the vein. In the former the intersection of the shaft and the vein is often a troublesome point and requires frequent attention and often retimbering, while in the latter position the support of the entire length of the shaft requires close watching and frequent repairing. Other things being equal, a shaft in the footwall entails the minimum and a shaft in the vein the maximum cost of maintenance. In all forms of vertical-shaft timbering there are certain common essential elements: The collar, the set, the studdles, the bearing timbers, the blocking, the hanging bolts, the lagging, and the lagging strips.

Collar.—The structure placed at and in the near vicinity of the mouth of the shaft is termed the collar structure. The simplest structure consists of a heavy set (the collar set) resting on heavy sills that extend across the shaft at the end and at each divider. The sills are extended from 4 to 8 ft. outside the timbers. The collar set is bolted to the sills and the first round of hanging bolts is attached. In firmer ground the sills are short, while in soft ground they are given greater length and cross sills used on the extensions in order to get greater bearing area. In more permanent construction a concrete wall extending from two to 10 sets deep is constructed outside the shaft timbers, and on this wall the bearers for the collar set are bolted. In Fig. 157, *a* shows the construction of a timber collar set supported by verticals and diagonals and intended to be filled in with waste rock; *b*, the use of end trusses and beams, and *c*, the use of transverse timbers supported on a concrete foundation.

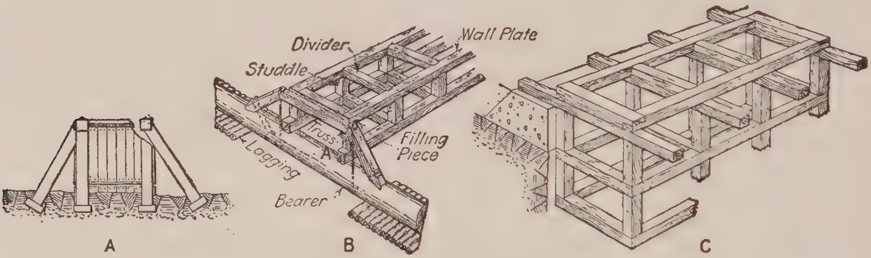


FIG. 157.—Construction of collar sets. (*a*, *A.I.M.E.*, *b*, *Bull.* 1; *Minn. S. of M.*)

In most cases the ground at the surface and for some distance below is disintegrated and as a consequence loose and heavy. The first 5 or 10 sets should be designed to meet this condition. This is accomplished by using heavier timbers or closer spacing of the sets.

Shaft Set.—The rectangular shaft set is the commonest structure used. It is made up of two wall plates, two end plates, and dividers that separate the shaft into two or more compartments. For firm rocks the members are 8 by 8 to 10 by 10 in., for less firm rock 12 by 12 in., and for heavy ground from 14 to 14 to 20 by 20 in. in cross section. The sets are usually placed 5 ft. apart and separated by the studdles or vertical posts. The joints between end plates and wall plates are a combination of square and bevel. The dividers are held in place by a dovetailed

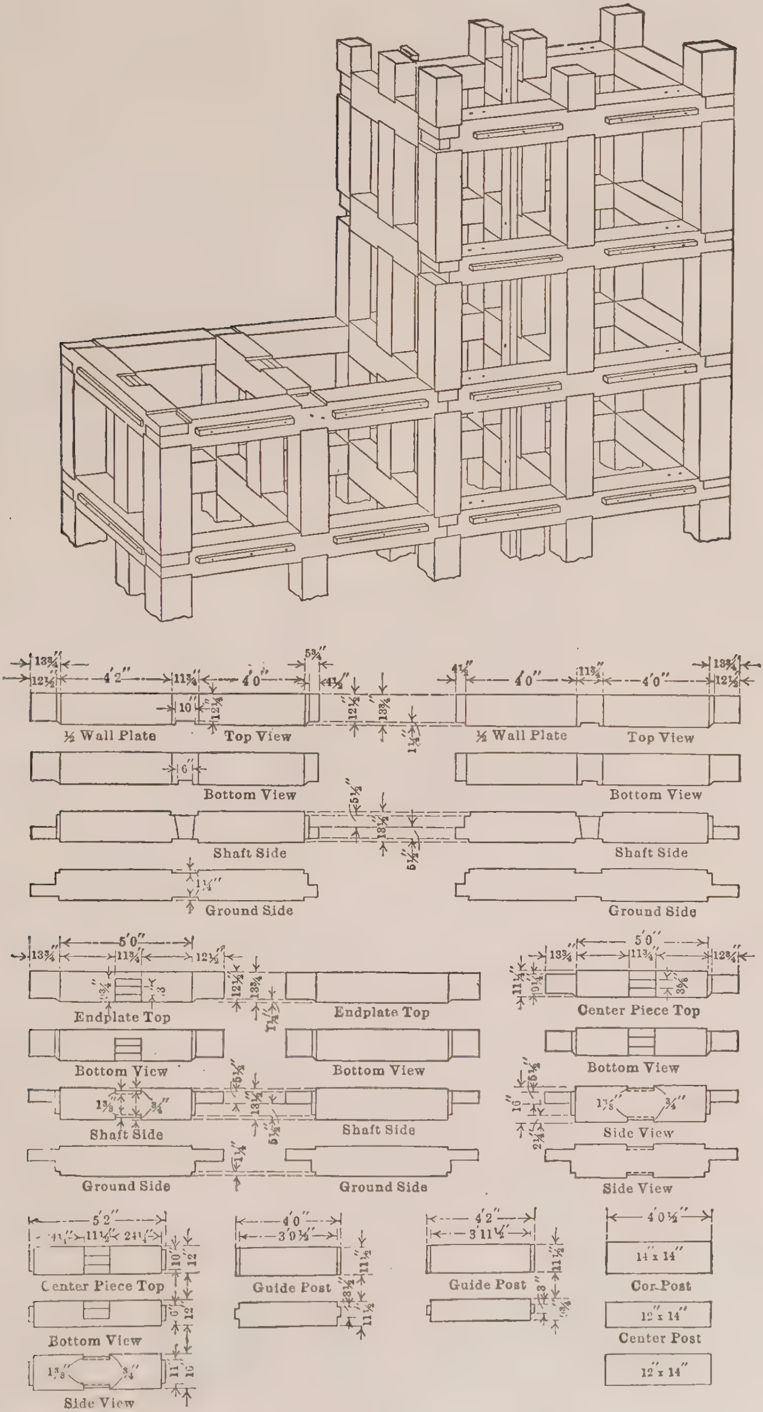


Fig. 158.—Four-compartment shaft timbering. (*Trans. A.I.M.E.*)

mortised joint. Figure 158 gives the details for a four-compartment shaft used in the Butte district. Wall plates are spliced where they are too long to be conveniently handled. A short splice is made at a divider. Dividers are usually of rectangular section, the longer dimension being equal to the dimensions of the wall plate. For example, the divider for an 8 by 8 wall plate would be 6 by 8 in., and for a 12 by 12, 10 by 12 in. Each wall plate is provided with two pairs of holes near the ends. These are used for the hanging bolts. In addition two shallow saw cuts are placed on the inside of the wall plates about an inch within the edge of the end plate. These are for the purpose of lining in the set. A long-fiber strong timber which does not rot readily is used for shaft sets. Oregon pine and yellow pine are frequently used.

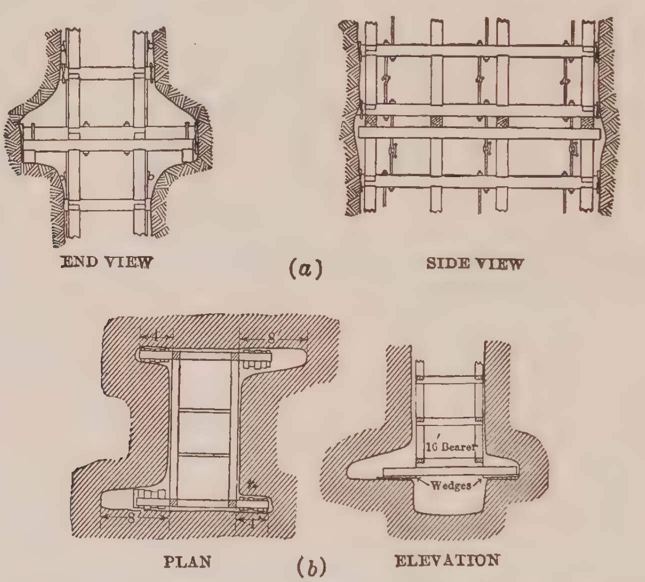
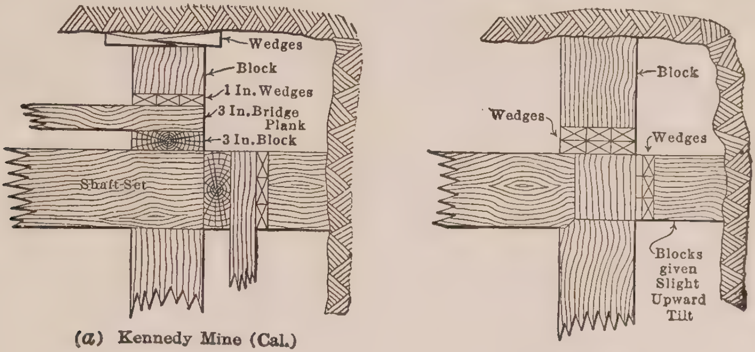


FIG. 159.—Construction of bearing sets. (*a, Trans. A.I.M.E., b, Bull. Minn. S. of M.*)

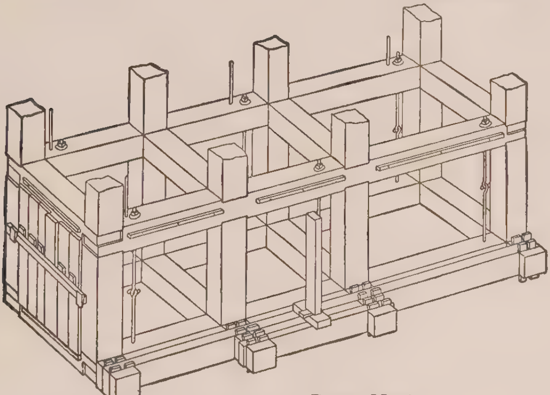
Studdles.—The vertical members separating the sets are placed at each corner and at each intersection of the dividers and the wall plates. In the Butte district they are also placed between the dividers at the center of the divider for the purpose of giving additional support to the guides. Studdles are mortised into the wall plates from ½ to 1 in. The ends are square. They serve the function of carrying the weight of the sets above the bearers and to stiffen the sets.

Bearers.—Cross timbers or bearers supported by hitches cut in the shaft walls support the weight of the shaft timbers. They are placed beneath end plates and dividers at intervals of 50 to 250 ft. or more. A common interval is 100 ft.; heavy ground requires a shorter interval than does firm rock. They are omitted in concrete-lined shafts where the bond between the concrete and the rough wall rock is considered sufficient. However, especially in the upper part of a shaft, enlargements are sometimes made in the wall rocks and the concrete is corbeled back into the walls so as to give an additional bond. Where wall rocks are weak a thickening of the shaft walls by increasing the rock section also serves to strengthen the structure. Wall bond is sometimes increased by inserting steel rods in inclined holes drilled into the walls. Figure 159 shows the construction of a bearer set. A bearer set is also shown in Fig. 166.



(*a*) Kennedy Mine (Cal.)

(*b*) Brakpan Shaft (S.A.)



(*c*) Butte, Mont.

FIG. 160.—Methods of blocking shaft timbers. (*c*, *Trans. A.I.M.E.*)

Blocking.—A shaft set is lined vertically with the set above and is held in place by wedges and short wooden blocks placed at the corners and opposite the dividers, as shown in Fig. 160. Steel sets are preferably blocked by concrete enclosed in sacks. These are puddled and jammed into place on inwardly sloping rock faces at the corners or are supported on projecting pieces of discarded drill steel set into 1- to 2-ft. drilled holes (Mather mine). Another method employed at the Mather mine consists of a wooden form hook-bolted to the flange of the set and braced from the stage below the set; the form makes an angle of 45 deg. with the wall rock, and concrete is puddled into place between the wall rock and the steel frame. In effect, short corbels of right-angled section are formed at the corners, or a ring corbel on all sides of the shaft set may be constructed.

Hanging Bolts.—Each set is suspended from the next above by four or more hanging bolts. Each is made in two parts of 1⅛- or 1¼-in. round iron. The two pieces are joined by hooks and the ends threaded. Cast-iron washers and nuts are used for the attachment to the wall plates. In placing a set the two wall plates are lowered and suspended from the hanging bolts. The end plates are then placed in position, being supported upon the horns of the wall plates. The set is then elevated almost into position by screwing up on the nuts of the hanging bolts; the dividers and studdles are placed, and then the lower set is screwed up. The set is now ready for blocking. The hanging bolts serve no further use after the sets are supported on the bearers, so that they can then be removed and reused, although they are often left in place to contribute to the general strength of the structure.

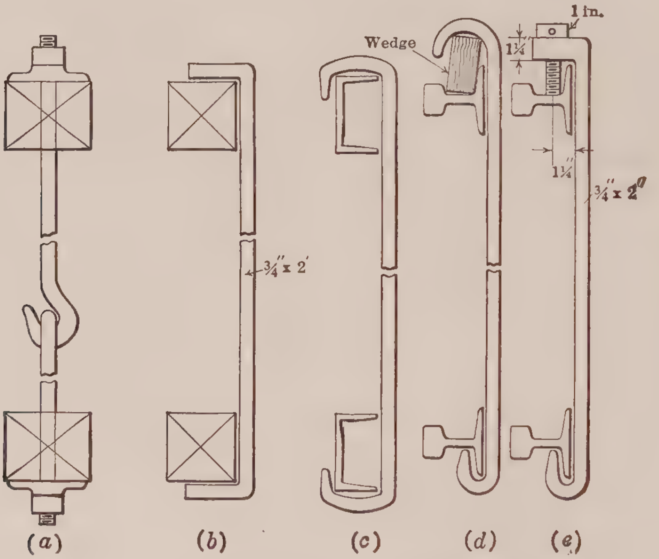


FIG. 161.—Hangers used in suspending shaft sets.

In Fig. 161 five types of hanging bolts are shown; *a* is the ordinary type, *b* is a type used in a German colliery, *c* is used with channels, *d* is a type proposed for use in the Pioneer shaft, Minnesota, and *e* is the hanger used in sinking a steel-supported shaft at the last-mentioned mine.

Lagging and Lagging Strips.—Two- to 3-in. lagging boards are used to prevent falls of wall rock. They are held in place by 2 × 2-in. strips nailed to the wall plates. Close lagging is usually required, but in firm rock spaces may be left between the boards. In other cases the lagging may be used only where the wall rock is shattered or weak. At the Hollinger mine, V-beam copper-bearing galvanized-iron sheets, 62 × 28 in., were used in place of lagging boards. The sheets were placed against a nailing strip and fixed into position flush with the inside of the shaft timbers by small metal clips, which were also nailed into position. At the Mather mine, 14-gage galvanized corrugated sheets were clip-fastened to the inside flanges of the H beams of the steel sets. Between the skip and ladder compartments $3/16$-in. lapped steel sheets, sealed by concrete with the rock walls, divided the shaft and provided a two-way ventilation system. The use of steel shaft sets, concrete blocking and lining at some places, and the corrugated wall sheets resulted in a fireproof shaft. A chrome-nickel-copper steel was used for the steel sets to reduce corrosion to a minimum.

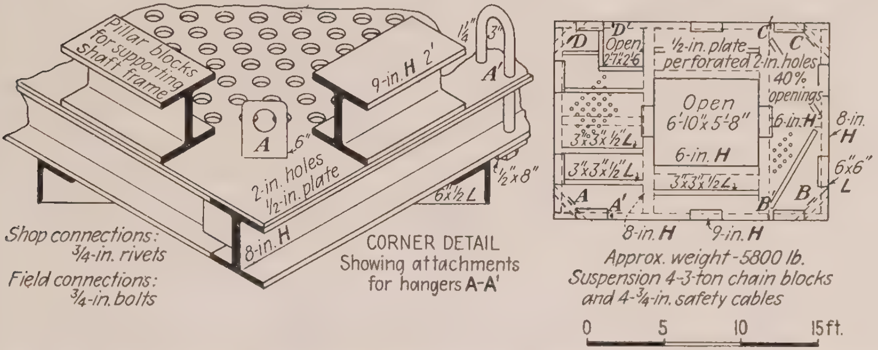


FIG. 162.—Steel-blasting bulkhead on right; enlarged detail of corner of platform on left, King mine No. 3. (*C.I.M. & M.*)

Blasting Boards and Steel Bulkheads.—Blasting in shaft sinking may damage the shaft timbers although the shaft sets may be some distance above the shaft bottom. Blasting timbers were first used to avoid serious damage. These were half-round timbers, attached by chains and tightened by wedges, upon the underfaces of the shaft timbers. The working platform or staging used for placing the shaft set in position finally developed into a suspended bulkhead which served for erection and also as a protection from the blasting effects. Such a bulkhead is constructed of 6- to 8-in. H beams and is covered with half-inch perforated plate through which an opening is left for the operation of the sinking bucket. The platform is suspended by four chain blocks for convenience in lowering and adjustment; four safety cables are also used. Design varies in detail. As an additional protection the lower faces of the steel frame are surfaced by half-round timbers held in place by U bolts. Figure 162 depicts a plan and corner detail of a bulkhead for a large shaft.

Guides.—A strong, straight-grained wood, usually Douglas fir, is selected for guides. Dimensions range from 4 × 4 in. up to 7¾ × 9¾ in.

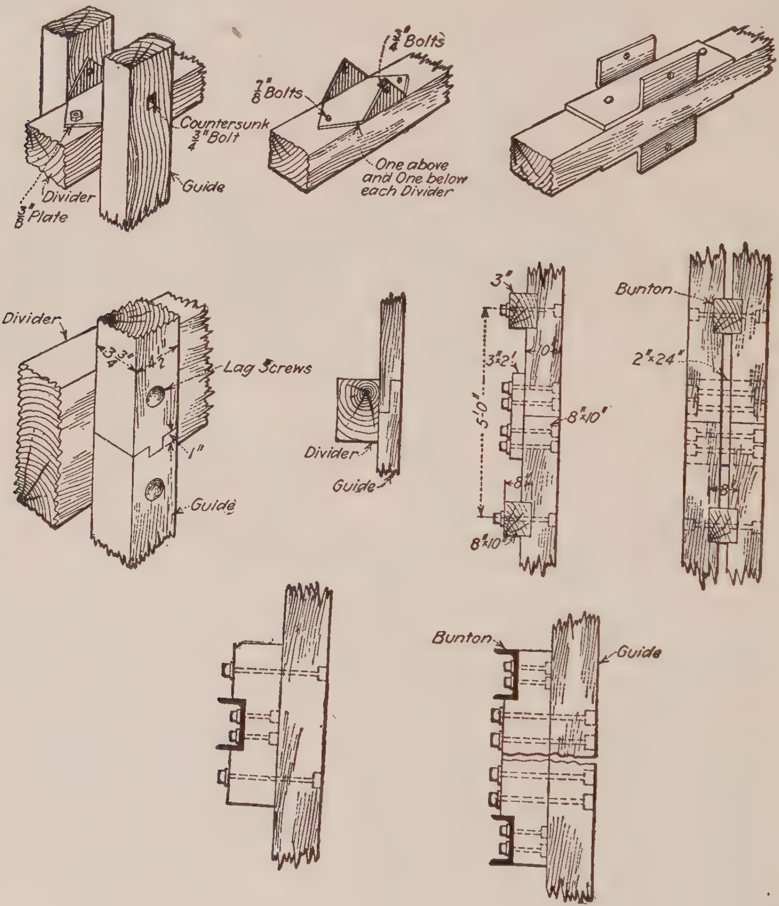


Fig. 163.—Construction of guides.

The use of safety dogs on cages necessitates a guide timber of ample cross section and great strength securely fastened to the timbers of the shaft set. Steel guides, such as ship channels, rails, and wire rope, are also used, but the wooden guide is preferred for cage operation. Steel rails are often used for skip operation. Various methods of attaching guide timbers to timber and steel members are shown in Fig. 163. In concrete-shaft construction cast-iron receptacles are placed at intervals in the walls of the shaft lining for the attachment of guide bolts.

Staging Used in Blocking and Lagging.—Shaft timbering follows excavation at a distance of 20 to 50 ft., and it is therefore necessary to construct a staging from which to block, line, and lag the set. The simplest method is to hang two additional wall plates below the set to be finished. Crosspieces (Fig. 164) are placed on the wall plate and on these two or three long planks. A tight floor of short planks placed across the width of the shaft is then constructed and the staging is ready for use. Still another method is to attach four chains to the hanging bolts above the last set. Through stirrups on the ends of the

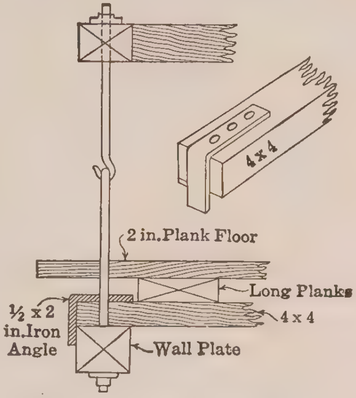


Fig. 164.—Shaft staging.

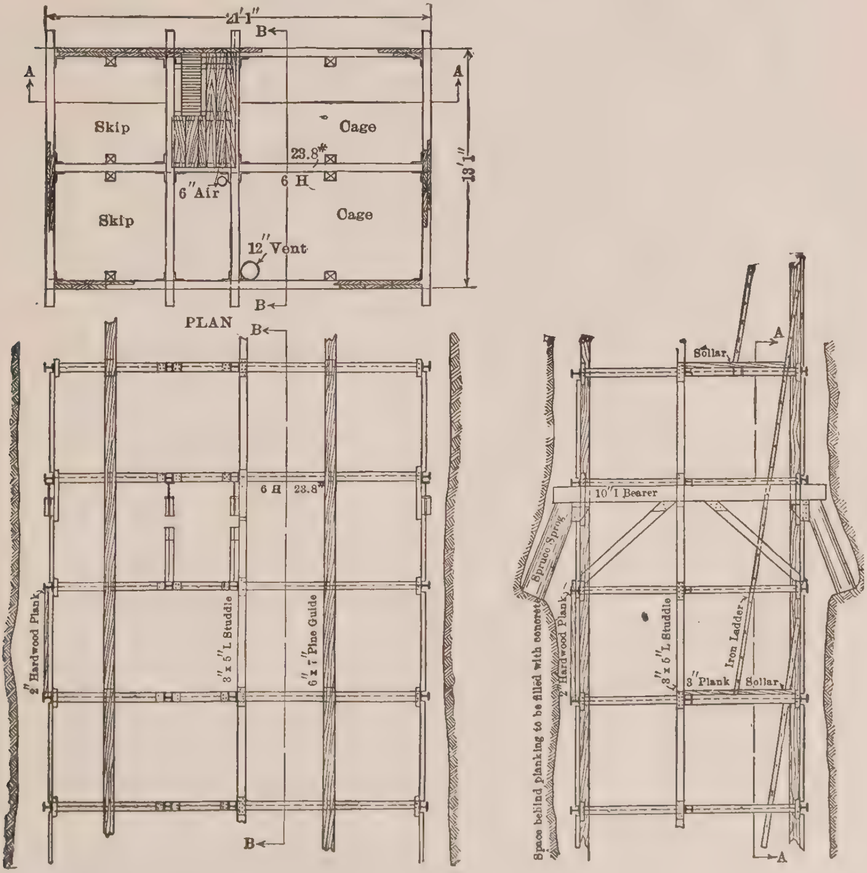


Fig. 165.—Steel-shaft set used in the Woodbury shaft. (*Trans. L.S.M.I.*)

chains, 4- by 4-in. timbers are inserted, and on these are placed the long planks. Then the floor is laid on the planks. Where shaft timbers are kept close to the bottom the timbermen work on top of the muck pile.

Steel Shaft Sets.—Where permanency is desired, where only unsatisfactory timber is available, or where timber decays very rapidly as in tropical countries, structural-steel shapes can be used in place of timber.

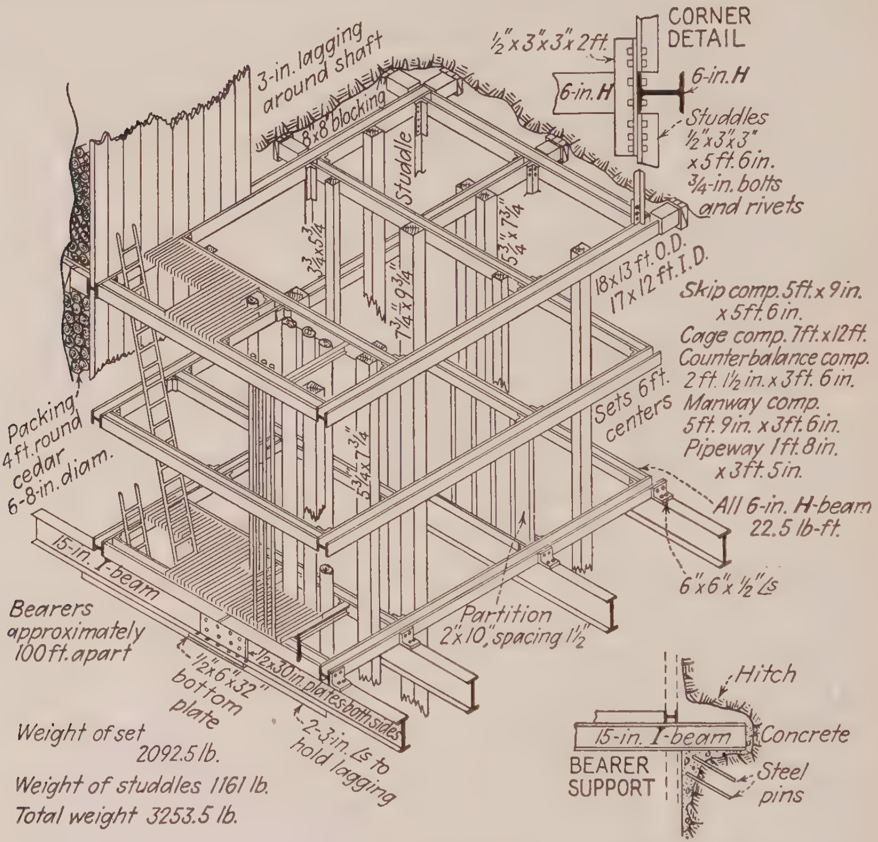

3-in. lagging around shaft
8x8 blocking
Studdle
½"x3"x3"x2ft.
CORNER DETAIL
6-in. H
6-in. H
Studdles ½"x3"x3" x5ft. 6in.
¾-in. bolts and rivets
3¾x5¼
5¼"x7¾"
7¼"x9¼"
18x13 ft. O.D.
17x12 ft. I.D.
Skip comp. 5ft. x 9in. x 5ft. 6 in.
Cage comp. 7ft. x 12ft.
Counterbalance comp. 2 ft. 1½ in. x 3ft. 6 in.
Manway comp. 5ft. 9in. x 3ft. 6in.
Pipeway 1ft. 8in. x 3ft. 5in.
Sets 6ft. centers
Packing, 4ft. round cedar 6-8-in. diam.
5¾x7¾
All 6-in. H-beam 22.5 lb-ft.
6"x6"x½"∠s
15-in. I-beam
Bearers approximately 100ft. apart
Partition 2"x10", spacing 1½"
½"x6"x32" bottom plate
½x30in. plates both sides
2-3-in. ∠s to hold lagging
Weight of set 2092.5 lb.
Weight of studdles 1161 lb.
Total weight 3253.5 lb.
Hitch
15-in. I-beam
Concrete
BEARER SUPPORT
Steel pins

Fig. 166.—Steel-shaft construction, King mine No. 3. (*C.I.M. & M.*)

The design closely follows that used for timbers. Bolts are used to fasten the members together. Fastening angles should be riveted to the members. A design for a steel shaft set is shown in Fig. 165. Steel shaft sets are constructed of H beams, I beams, angles, or rails.

A later type of steel shaft construction than the foregoing is shown in Fig. 166. Figure 167 shows the concrete collar construction used in the upper part of the shaft.

Reinforced-concrete Sets.—The substitution of reinforced-concrete members in place of timber in shafts of rectangular section involves no structural difficulties, and while the weight of the members is an objection, their handling involves no great difficulty. Where concrete is used, thin reinforced slabs can be used for lagging. The concrete structure, like the steel, has the obvious advantage of being fireproof.

Masonry Support.—Concrete and brick are used in shafts of circular or elliptical section for obvious structural reasons. In putting in this form of support a temporary support of steel or timber is put in for a distance ranging from 50 to several hundred feet, and then the concrete or brick is started from a bearing ring. The masonry is put in sections from the top downward, each section being constructed from the lower bearing ring upward to the next above. Bearing rings are of two general types: One consists of a number of steel rods (2 in. in diameter) set in holes drilled in the rock and supporting a mantel plate on which the masonry is started; and the other is of the nature of a ring corbel started outside the shaft section. The use of steel rods is possible only in firm rocks. A comparatively shallow shaft can be completely excavated and

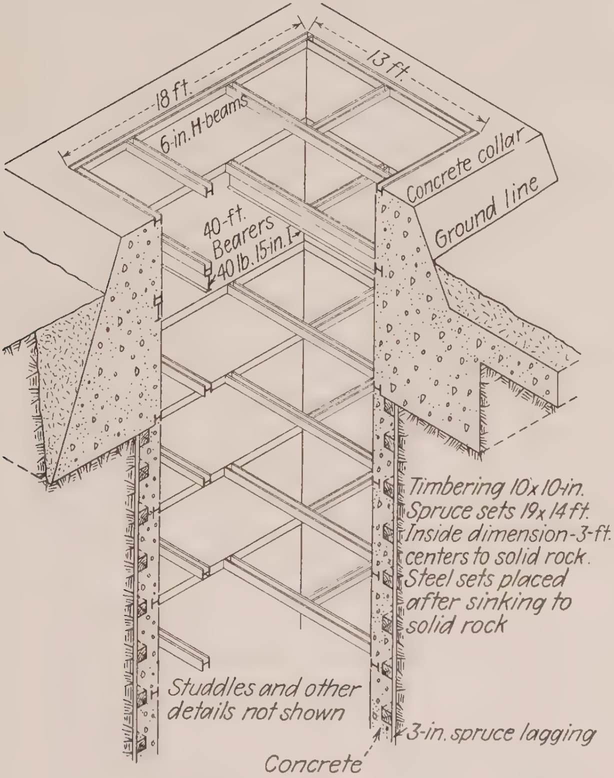


FIG. 167.—Collar construction, King mine No. 3. (*C.I.M. & M.*)

temporarily supported by timber or light structural steel; the masonry lining can then be started at the bottom and continued to the top.

Details of Concreting.—The mixtures used in shaft concreting are 1.5–2–5, 1–2–5, 1–3–5, and 1–3–6, the mixture in the proportions of 1 of cement, 2 of sharp, clean sand, and 5 of broken rock being probably used more than the others. A compact rock is chosen and the size limited to 1.5 or 2 in., the former dimension being preferable. Mixtures are made on the surface with some form of concrete mixer. A half-yard mixer is best where the concrete is lowered in buckets; where pipes are used a larger mixer is more economical. The concrete is mixed wet, lowered in a bucket, skip, or bottom-discharge hopper, and discharged into a wide chute that delivers it to the forms. With the bottom-discharge hopper the concrete is discharged through a flexible spout attached to the hopper. The best method is a pipe which extends from a hopper at the mixer to the forms. The end of the pipe is attached to a flexible spout, by means of which the stream of concrete can be directed into any part of the form. A 4-in. pipe and a 6-in. flexible spout are adequate for shaft work. Tamping is not often resorted to, but in its place the concrete is worked down against the forms and the walls by long-bladed shovels.

The forms used are of the collapsible-steel type. The form for a circular shaft is in four sections and is constructed of steel plate, 10-gage plate or plate ⅛ in. thick, angles, and channels. It is thoroughly braced. On each side of one diameter a wedge-shaped piece of timber 5 in. thick divides the two halves and is held in place by angles and bolts. By knocking out the two wedges the steel forms can be readily removed. The form is 5 ft. high and ordinarily 10 forms are used. The lowest form is supported on timbers placed in hitches in the walls and lined into position. A flooring is constructed on the timbers beneath the forms and is given a 45-deg. pitch toward the wall of the excavation. The purpose is to bevel the lower part of the first concrete ring so that the joint between it and the next section below can be readily made. In some cases the first form is constructed with an inner metal apron that forms the bevel. Forms are left in place at least 72 hr. Where concrete is placed in cold weather the aggregates are heated with steam jets and warm water is used in mixing. The mixed concrete should approximate 140°F.

In the concreting of the Campbell shaft, Warren, Ariz., steel forms were used, ¼-in. plate stiffened by angles and hinged at the corners to facilitate handling; the height of each section was 6 ft. 8 in., so that three sets of forms take the height of a 5½- by 7½-in. by 20-ft. guide timber. Concreting started from the bottom of the shaft, the timbers being stripped from a section as soon as a set of forms became available. The

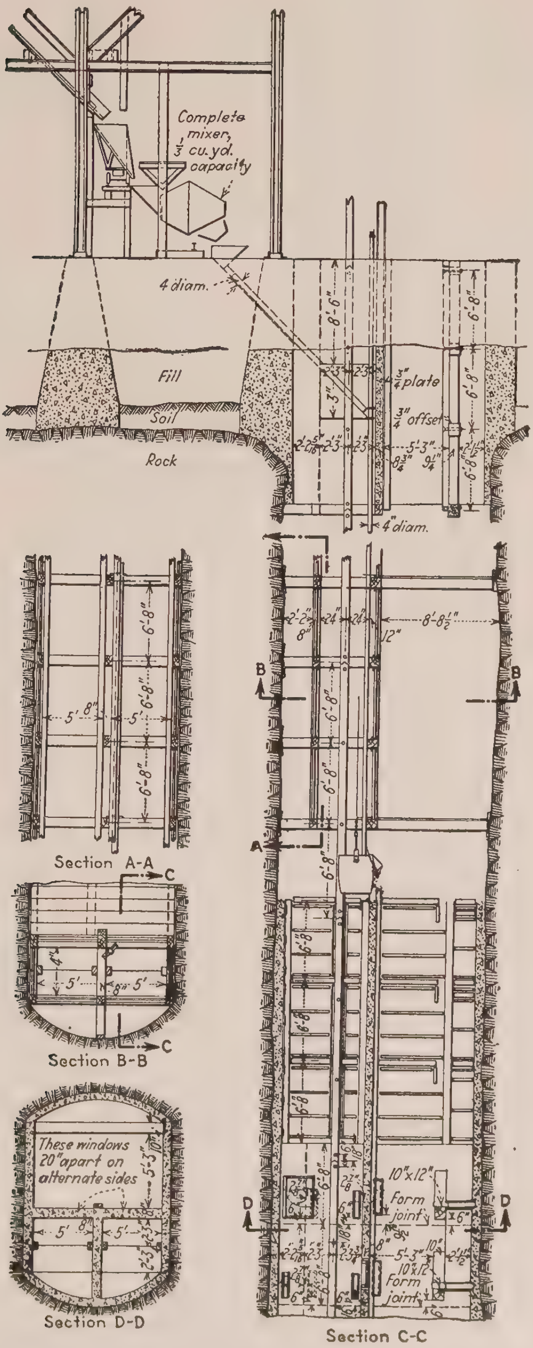


Fig. 168.—Cross section of Campbell shaft. (*Eng. Min. Jour.*)

section of the shaft is shown in Fig. 168, which also shows the layout of the concreting arrangements. The forms and receptacles for the securing of the guide timber bolts are shown in Fig. 169. A ½-cu. yd. mixer was used, and material bins were so placed as to enable the sand, crushed rock, and cement to be handled by a transfer car above the mixer, which discharged into a hopper connected with a 4-in. pipe leading down the shaft. The lower edge of a form is placed 18 in. above the junction of a guide timber, and the upper edge of three forms in position projects 18 in.

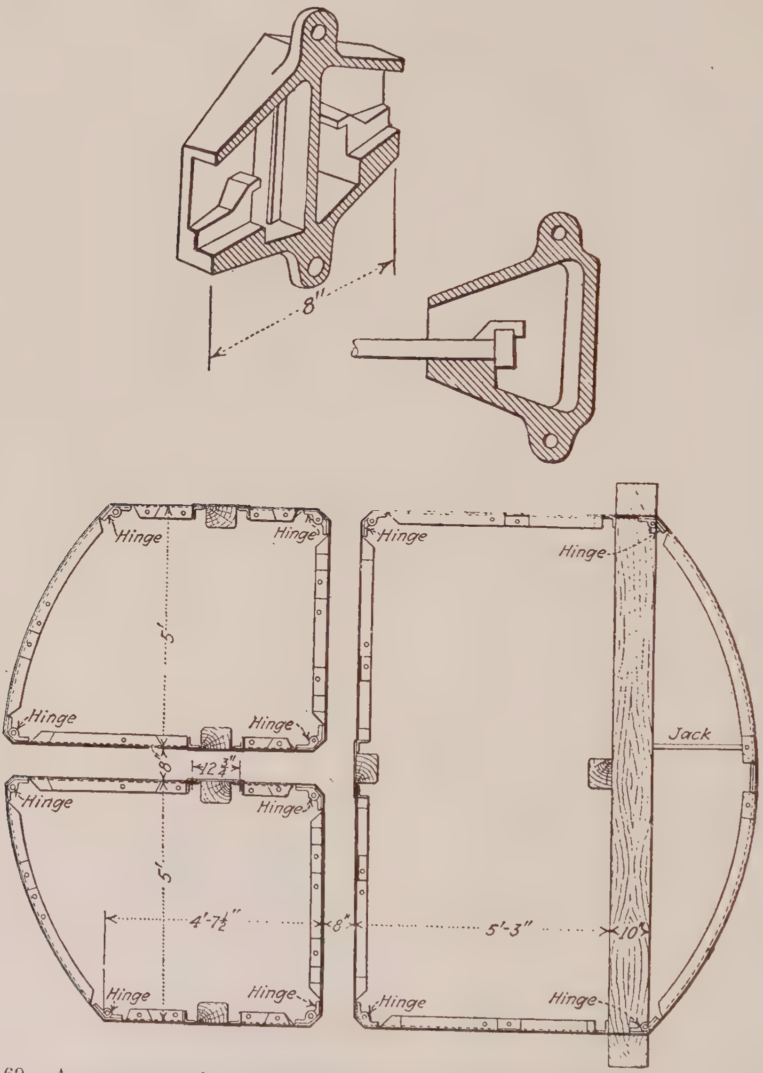


Fig. 169.—Arrangement of steel forms used in concreting the Campbell shaft; receptacles used for guide bolts.

above a junction. Guide timbers are lined in place and, since the edges of the forms abut against the sides of the guides, the closure of the space to be filled with concrete is accomplished. In the skip compartments, the forms on each side of a corner are braced by bars, the bent ends of which are placed in holes in the top of the angle reinforcement. The bars, being of fixed length, place the contiguous parts of the form in exact position and also act as braces. A similar arrangement is used in the cage compartment, the two forms at the curved ends being supported by a special jack.

Openings or "windows," 20 ft. apart and on alternate sides of the guide of the cage compartment, are left in the dividing wall separating the cage and skip compartments. In addition, windows are left in the partition between the skip compartments on each alternate form. The windows are 30 in. wide by 42 in. high. Sockets (scaffold boxes) are left in end walls and partition walls of the skip compartments, on a line 6 in. above the lower edge of each form. These sockets are 26 in. high, 7 in. wide, and 6 in. deep. Sockets are also placed in the walls for the 10- by 12-in. divider in the cage compartment and are on a line with the sockets in the skip compartments. They are 48 in. high, 10 in. wide, and 6 in. deep. A horizontal groove, 3 in. wide at the bottom, 4 in. deep, and 6 in. wide at the inside edge, is placed in the curved section of the cage compartment at a point corresponding to the top of the divider, the groove being formed by a wooden piece bolted to the steel form. The sockets are used in connection with the concreting operations in the cage compartment as receptacles for supporting timbers, upon which a platform is carried beneath the three sets of steel forms. After a top form is in place, a platform is constructed and supported upon the reinforcing angles of the forms; the forms are filled and tamped; and then the shaft is stripped for the next form above. Sockets are placed in the skip compartment for emergency use after the shaft is in operation. The crossheads used in the skip compartments are so constructed as to clear the forms in place. Forms to be moved are taken off, set on the crosshead platform, hoisted to the proper elevation, and placed in position.

Two 8-hr. shifts separated by 4-hr. intervals were used in concreting, a six- to eight-man crew being required and two men during the 4-hr. interval for placing braces and making ready. The equivalent of two rings, or 13 ft. 4 in., was placed in 24 hr. A 1-3-5 mix was used; minimum wall thickness 6 in., maximum 3 ft., and average 18 in., depending upon the amount of overbreak in the shaft section.

Concrete blocks have been used for circular shafts and precast concrete slabs in rectangular shafts. W. V. DeCamp made use of concrete rings for the United Verde shafts. The idea of the arrangement is shown in Fig. 170. DeCamp estimated comparative costs of $127 for the ring method as against $215 per foot for a continuous concrete lining. E. H. Clifford described a similar arrangement employed at the City Deep mine, South Africa; the shaft in question is 20 ft. in diameter and is provided with concrete rings at intervals of 10-ft. centers for the support of shaft guides, pipes, and power cables. Each ring is 18 in. deep and 15 in. wide and is cast about 20 steel pegs, 24 in. in length, driven into drill holes in the walls at each ring. Clifford credits the arrangement to E. C. J. Meyer.[1]

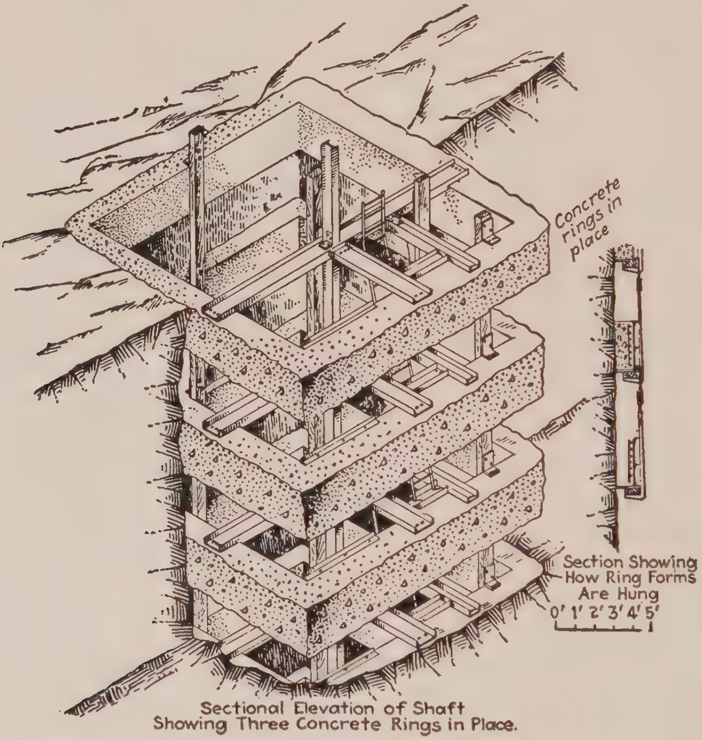

Concrete rings in place
Section Showing How Ring Forms Are Hung
0' 1' 2' 3' 4' 5'
Sectional Elevation of Shaft Showing Three Concrete Rings in Place.

FIG. 170.—Ring method of shaft support. (*Eng. Min. Jour.*)

Shafts in Loose Ground.—Where loose ground is encountered forepoling is resorted to and timbering follows the excavation without an interval. The poling boards are chisel-shaped and driven at an angle outward. The close timbering required often necessitates the dividing of long wall plates so that they can be handled in the restricted space.

Shafts of small cross section are also supported by cribbing. The cribbing boards are cut as shown in Fig. 171. Boards 2 by 6 in. or 3 by 6 in. are commonly used. The advance is made by cutting a trench on

[1] See *Eng. Min. Jour.*, vol. 121, p. 329; *Trans. I.M. & M*, vol. 30, p. 247.

the side and forcing the board down into position. After placing four boards the material in the center is removed.

Shafts in Extremely Soft and Loose Ground Containing Water.—Wooden or masonry caissons are used, and in some cases the pneumatic caisson is resorted to. The caisson is a cylindrical structure provided with a cutting shoe on its lower edge. It is forced down by its own weight, by loading with weights, or by jackscrews as the excavation proceeds, the shoe being at or in advance of the bottom of the pit. The resistance to the sinking of a caisson is approximately proportional to the area in contact with the walls. The caisson may be likened to an enormous well casing amply strong enough to support the walls of the excavation and carried along with the excavation. The pneumatic caisson is used in very soft material saturated with water. It differs from the open caisson in that an airtight working chamber is constructed in the lower part of the caisson. Compressed air is used in this chamber under sufficient pressure to exclude the water. An air lock enables access to the working chamber to be obtained without interfering with the air pressure.

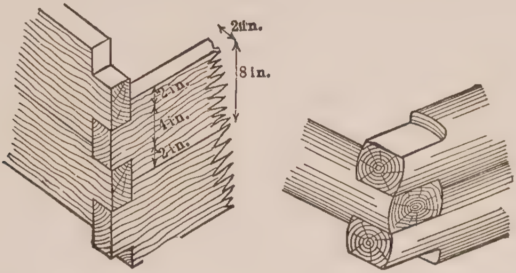


Fig. 171.—Shaft cribbing.

Both the open and compressed-air caissons are available for comparatively shallow shaft sinking from the surface. About 125 ft. may be considered the depth limit practicable for both systems. An alternate method consists of the driving of steel sheet piles and the removal of the inclosed material by dredge-bucket excavation.

Shaft Support Where Water Must Be Sealed Off.—The most effective method of supporting a shaft and sealing off water at the same time is by the use of cast-iron tubbing. The shaft, where excessive water is present, must be sunk by special methods such as the Kind-Chaudron or freezing methods. The excavation is sunk to an impervious formation and the lining of cast-iron tubbing rings is inserted. Tubbing may be likened to sectionalized cast-iron pipe of large diameter (up to 18 ft.). Where the shaft cannot be drained a "moss box" is used on the lower end of the tubbing and the tubbing is lowered, the sections being placed on the top of the "pipe," which is suspended by heavy cables. In order to reduce the weight a diaphragm is placed just above the moss box, and connecting with this at the center is a pipe in which the water displaced is free to rise. The tubbing can be considered as a hollow air-filled vessel lowered into the excavation. The buoyancy of the tubbing reduces the weight that must be suspended. By running water into the tubbing above the diaphragm an excess of buoyancy can be overcome. The tubbing is lowered into place; the moss box in closing forces the mat of moss against the sides of the excavation and makes the water seal. The space between the tubbing and the sides of the excavation is filled with concrete lowered in special buckets from the surface, and the shaft is then ready to be pumped dry.

Where the shaft can be kept drained, the tubbing is started from the bottom. First a wedging curb or ring of cast iron is placed and a watertight joint is formed by driving wooden wedges in the space between the wall of the excavation and the outer surface of the curb. The successive rings of tubbing are bolted in place. Tubbing rings are made either corrugated or with smooth exteriors and ribbed interiors.

CHAPTER XI

SUPPORT OF MINE WORKINGS—(*Continued*)

PILLARS

One of the most important vertical elements of support is the pillar of ore, mineral, or coal left in place. It is used particularly in coal mining. Shaft bottoms are protected by leaving a relatively large area of unexcavated coal about them. Entries and haulageways are also protected by pillars usually 40 to 50 ft. in width and paralleling the length. Side entries are separated by chain-pillars, 12 to 20 ft. in width. In mining the coal, rooms separated by pillars are driven and the pillars mined out after completing the rooms. In metal mines where shafts are driven in the ore body a portion is left on either side of the shaft for its protection. The shaft pillar is from 25 to 50 ft. in width. In mining wide ore bodies the ore body is divided into blocks, and alternate blocks are left as pillars until the intervening blocks are mined out.

Vertical ribs separating stopes along the length of the strike of the ore body are frequently left for the purpose of preventing any general movement of the walls until the stopes are mined out and filled. Arches are left in wide ore bodies between the top of the stope and the level immediately above for the same purpose.

Mining practice has no widely accepted rules as to the proportions of pillars. In coal mining, shaft pillars are frequently left of a diameter equal to the depth of the shaft. In a very deep seam this is undoubtedly excessive, while in a shallow seam insufficient protection of the surface would result from such a practice. In mining the coal, the proportion of coal left in pillars ranges from 30 up to 85 per cent. The tendency of modern practice is to win the greatest amount of coal on the second working, and as a consequence a large proportion of coal is left in pillars. Theoretically, the area of the pillars should be such that with a moderate factor of safety, say 2 or 2.5, the entire weight of the superimposed strata would be supported. The superimposed stratum is cantilevered over the mined area and also acts partly as an arch. Thus only a part of the weight of the superimposed stratum need be carried. Just how much is problematical, and as a consequence the above rule would leave too large an area in pillars. Other considerations, particularly that of safe mining, lead to the use of a relatively large pillar area instead of reducing it to the limit demanded by the strength of the materials involved. Douglas

Bunting has discussed the proportioning of pillars from theoretical considerations and his equation has been given in another part of this chapter.

In metal mining still fewer theoretical specifications are to be found. The pillar proportions can be roughly calculated from theoretical considerations, but this is by no means always done. In general, pillars in narrow veins should have a width equal to the thickness of the vein, while in wide veins a width ranging from one-half to one-fourth the width of the vein can be taken.

Rock packs are substituted for pillars where the waste material is suitable and where the value of the ore won by their use is more than sufficient to pay for their construction. They are constructed of dry walls 2 to 4 ft. thick enclosing an area and filled with waste rock. For stability they are constructed of a minimum horizontal dimension equal to the height. Unlike a pillar they yield to a moderate extent as they take weight and shrink in a direction parallel with the load in approximate proportion to the load. Tests made upon rectangular and circular piers of mine rock gave the following results:

TABLE 70[a]

	Net tons per sq. ft. required to produce compression of per cent					Compression and load at end of test compression load	
	3	5	10	20	30		
Rectangular piers........	0.8	1.4	2.7	9.5	23.0		
Circular piers..	3.5	5.67	11.0	22.0	38.5	31.0	42.5

[a] *Bull. 25, U.S. Bur. Mines*, p. 55.

It is obvious that the care taken in the construction of "packs" or rock piers would determine the relation between compression and loading. The experimental test serves to give a general understanding of the action of such structures. Hepplewhite advocates the construction of packs in long-wall coal mining upon soft foundations, as it gives opportunity for consolidation without buckling the walls.

Timber Cribs.—Timber cribs are constructed by laying two square or round timbers on the ground at a convenient distance apart and placing two other timbers across the ends of the first two. The timbers are of the same dimensions. The operation is repeated until the desired height is reached. Cribs may be constructed square, rectangular, or triangular in section. The minimum dimension as compared with the height varies but may be taken as ranging from one-quarter to one. Cribs may be left hollow or filled with waste or may be constructed solidly of timber.

The solid timber crib as compared with the hollow timber crib develops a much greater resistance for a given percentage of compression. In Fig. 172 a timber crib constructed of 12- by 12-in. Oregon pine timbers 4 ft. square and 10 ft. high has been figured for different loads and corresponding compressions. In a structure of this type the wood is subjected to compression across the grain and as a consequence yields to a much greater extent than would be the case in end compression. In the example figured, a solid-timber crib would show 4 times the load for corresponding compressions. With the timbers notched so that the timbers form a solid outer wall, in the example given, the resistance is 3 times that afforded by the open structure for a corresponding amount of compression.

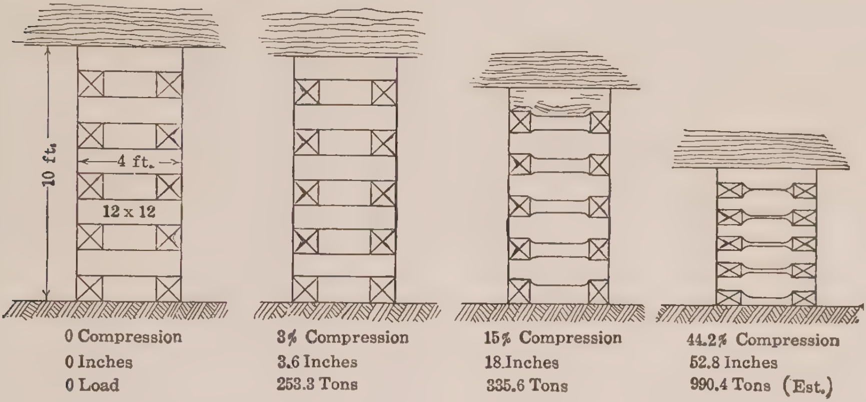


FIG. 172.—Timber crib showing compression under various loads.

The construction of a waste-filled timber crib requires that flat stones be wedged between the open spaces between the timbers forming the wall of the crib. The stones cause the compression of the whole area of the timber instead of merely the areas where the timbers intersect. Close-walled, waste-filled timber cribs are constructed by notching the timbers where they intersect sufficiently to cause the timbers to touch along their whole length. This results in a structure resembling a log cabin. The results of an experiment upon a timber crib filled with mine rock give the relation between amount of compression and loading.

TABLE 71.[a]—NET TONS PER SQUARE FOOT REQUIRED TO PRODUCE COMPRESSION OF PER CENT

	3	5	10	20	30
Timber crib filled with mine rock..	0.6	1.37	5.11	20.3	31.5

[a] *Bull. 52, U.S. Bur. Mines*, p. 54.

In coal mining timber cribs are used at the intersection of main entries and cross entries, at the intersection of gates with long-wall face, at room openings, and at long-wall faces. In metal mining they are used in stopes and at shaft bottoms where excessive ground pressures must be met, and in square-set stopes where sets begin to "jackknife" or get out of alignment. In all cases their elasticity renders them most advantageous as compared with masonry structures. The time element is greatly increased as compared with the use of rigid structures. Equivalent names for timber cribs are cogs, chocks, pigsties, and cording.

Props.—The prop is one of the most frequently used elements in support. It is subjected to compressive stresses. Structurally, it may be considered as a column fastened at one or both ends. The ratio of unsupported length to least diameter of side dimension ranges from 10 to 16. The practice in proportioning struts may well be studied, as it gives the engineer a quantitative idea of the loads that can be safely carried. It should be remarked that mine props are used up to the limit of their compressive strength and when they fail they are replaced. It is expected that they will last only a relatively short time. Various expedients are resorted to to prolong the period during which the load develops to the full supporting power of the prop. Headboards where a thick piece of timber is compressed across the grain are used more frequently than any other means. The prop most used is of round timber varying from 3 in. to 12 in. in diameter and from 3 ft. to 12 ft. in length. Various substitutes have been invented and used. Cast iron, cast steel, structural steel, wood and steel, and reinforced concrete are the materials which have been tried out, but practice has not definitely shown that any one of these types meets requirements sufficiently to be generally adopted or to displace the wooden prop.

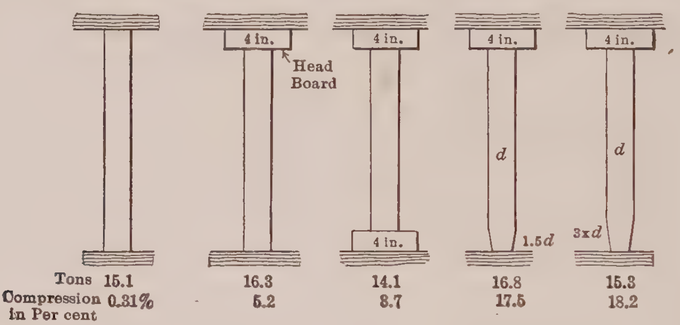
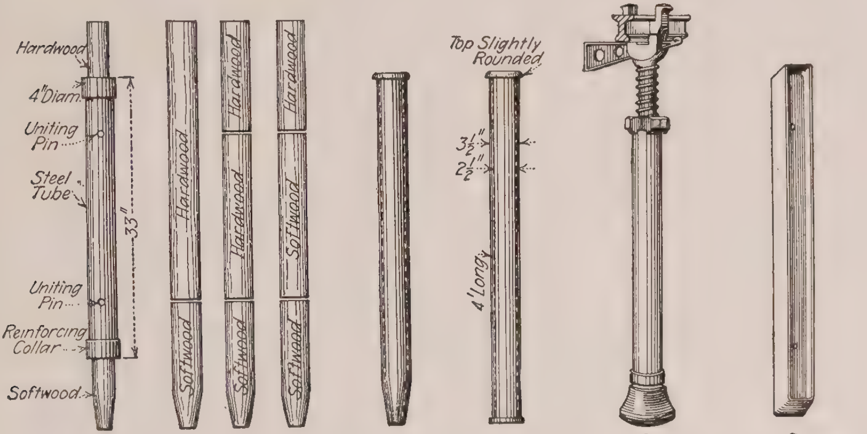


FIG. 173.—Loads and compressions, mine props.

The action of the wooden prop is illustrated in Fig. 173. The straight wooden prop without headboard admits of but very slight compression before failure. By the use of headboards 4 in. thick a much greater range of compression is available and in conjunction with a footblock of equal thickness a still greater compression is possible without failure. By tapering the foot of the prop and the use of a headboard greater compression is obtained. The tapered or Hepplewhite prop gives the maximum compression. Hepplewhite states that a 6-in. prop, 6 ft. long with a tapered portion 15 in. in length, will begin to burr at 15 tons weight and at 34 tons weight will buckle. It allows a roof depression of 15 in. The tapered portion of a prop is made equal in length to 2 to 2½ times the diameter; the end is made blunt, of a diameter approximately equal to one-half the diameter of the prop. Tapered props are used in supporting rooms in coal mining and to some extent for the posts on three-piece entry sets. Their main purpose is to prolong the life of the timber and reduce the maintenance.



A
Kirkby Prop

B
Hollow Cast Steel Tapered
Cast Iron Spencer Prop

C
Adjustable Prop

D
Pit Prop (*English*) H-Section

FIG. 174.—Steel-wood and metal props.

Probably one of the best steel-wood props is the Kirkby prop, shown in Fig. 174*a*. This consists of a light steel tube reinforced at the ends by bands. Various combinations of wooden plugs are inserted in the tube and held in place by pins. The projecting foot plug is tapered. Both foot- and headboards can be used. The range of compression equals that of the wooden prop. The main advantage of this prop is that pieces of timber too short to be used as props can be utilized with the steel tube. The prop is readily adjustable to length. It indicates the intensity of the pressure by the burring of the end. It can be reset a number of times and is almost as light as a wooden prop. Its principal disadvantage is the preparation of the timber pieces where the prop timber is irregular.

Steel tubular props made of old boiler tubes, the top filled with sawdust, disks of wood, or slack coal and the leg portion of solid iron or steel (acting like a plunger) have been tried and found satisfactory in some mines. Tubular props filled with concrete and closed by a tapered wooden plug have also been found satisfactory.[1]

Hollow cast-steel tapered props of the Spencer type can be used where the floor for a depth of a foot is moderately hard. This allows the prop to penetrate and thus serves the same purpose as compression in a wooden strut. If the bottom is too soft or too hard the prop is not satisfactory.

Spencer discusses the use of cast-iron props (see Fig. 174*b*) and states that they can be satisfactorily used where head- and footboards are used with them. He states that the bottoms of the props were easily broken by hammering. The greater weight and cost of both steel and cast-iron props preclude their use except where they can be readily retrieved and used over again. Their lack of convenient adjustability as to length is a serious objection. Hepplewhite states that it is very rare that cast-iron props break from vertical pressure.

There are also adjustable props consisting of two steel tubes, one telescoping within the other, and held by a clamp that admits of yielding when a load of 15 tons or more is reached. Hepplewhite states that some difficulty is experienced in setting the clamps tight. By the use of a long-handled wrench the clamp can be loosened and the prop recovered without danger to the worker. He states that good results have attended their use at gate ends and in conjunction with tapered props.

Figure 174*c* shows another English prop of the adjustable type. Many adjustable "pit props" have been devised by the Germans; the reader is referred to Heise and Herbst, "Bergbaukunde," for examples.

Hepplewhite describes the construction of reinforced-concrete props.[1] The mixture used was 1 cu. ft. ashes, 2 cu. ft. breeze dust (¼-in. mesh), and 1 cu. ft. cement. The reinforcement consisted of nine unwound strands from a ¾-in. steel-wire rope. Both pouring and tamping were used. The crushing strength of such a prop was 2,400 lb. per sq. in. The concrete prop offers possibilities of development. It is true that it chips and is not so convenient to handle as the wooden prop. It is also inelastic and must be used with head- and footboards. Hepplewhite states that concrete props are best alternated with wooden props.

Structural-steel props of H-beam section are used to some extent, although they are more suitable for permanent haulageways and shaft bottoms than for working faces. Figure 174*d* shows an English steel prop made from an H section.

In South African stopes of low inclination, concrete "pancakes" or disks of concrete 30 in. in diameter by 4 in. thick, with a 4-in. hole in the center to facilitate handling, are used to form a built-up column for supporting low stopes. The concrete is 1:3:7 of washed sand and 1½-in. quartzite rock, reinforced with strands of wire rope and cast in molds. The pancakes weigh about 250 lb. each. After seasoning they are placed in position upon a concrete foundation and stacked up to within 1 or 2 ft. of the hanging wall. The intervening space is filled with short lagging wedged tightly against the hanging. When they are placed upon a sloping footwall a thin layer of cement or sand is interposed between the separate layers to hold them in position until the top can be wedged in place. They have been widely used upon the Far East Rand.

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[1] See Hepplewhite, Substitutes for Wooden Supports. *Iron and Coal Trade Rev.*, Dec. 18, 1914, p. 768.

Stulls.—The stull involves essentially the same structural principle as the prop. It is, however, used as a horizontal or inclined member and as such frequently also has to serve the function of a beam in supporting working floors. The same proportions that were described for props hold good. For a very wide span an unbraced stull would of necessity have to be of large diameter, and a limit width of span is speedily reached by the size of the available timber. With bracing (vertical or inclined members), stulls of considerable length and relatively small diameter can be used; but it is obvious that we have in this case a near approach to the square-set system, which is preferable for wide ore bodies. Stulls where braced can be used with headboards in a manner described in a succeeding section. Where unbraced they are socketed in hitches cut in the walls. Where inclined, the foot of the stull is placed in a hitch and the head is headboarded and wedged. Headboards are essential where the loads are excessive for the size of stull. Where the stull is sufficiently strong to meet the side pressure, head- and footboards are unnecessary.

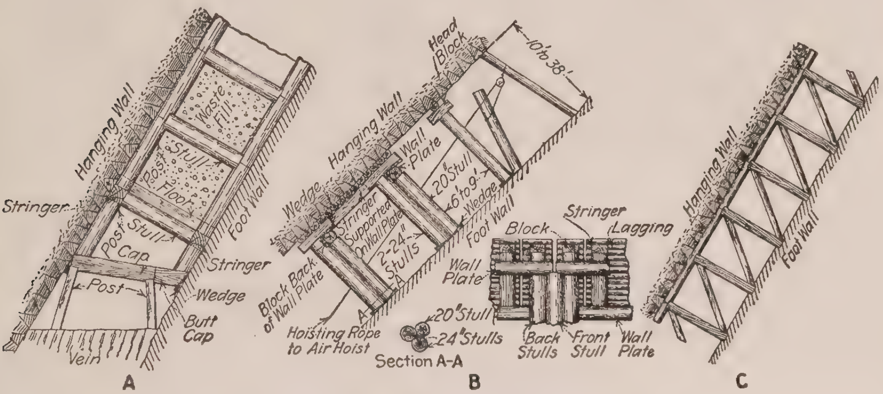

Fig. 175.—Stull timbering.

Where side pressure is very moderate, the chief function of the stull is to support the working floors. So far as my observation goes, tapered end stulls are not used, but this principle is worthy of trial and their use would obviate the headboards. Three different methods of stull timbering of stopes are shown in Fig. 175.

Square Sets.—The square set is composed of a cap, a girt, and a post. These members meet so as to form a solid 90-deg. angle. They are so framed at the intersection as to form a compression joint and join with three similar members. Thus the system is formed by the intersection of pairs of sets. An open framework composed of vertical and horizontal members at right angles is the result. This framework is built up and extended as the excavation of the ore proceeds. Each block of ore of the approximate dimension of a set is replaced by a set, which is temporarily blocked and wedged fast. The first tier of sets is termed the "sill floor," and these are usually started on a foundation of long timbers called "sills." The succeeding tiers are numbered consecutively, first floor, second floor, etc. The square set is usually 5 to 6 ft. square and 7 to 8 ft. high. The sill floor set is usually 9 ft. high. Round and squared timbers are used, although preference is given to the round timbers. Probably the least dimension is 10 in. round, although I recall one instance of 6-in. square timber having been used. The maximum-sized timber ranges from 12 to 16 in. The present tendency of mining practice is to use a 10-in. round timber and fill the sets with waste, keeping at most only two floors open near the face.

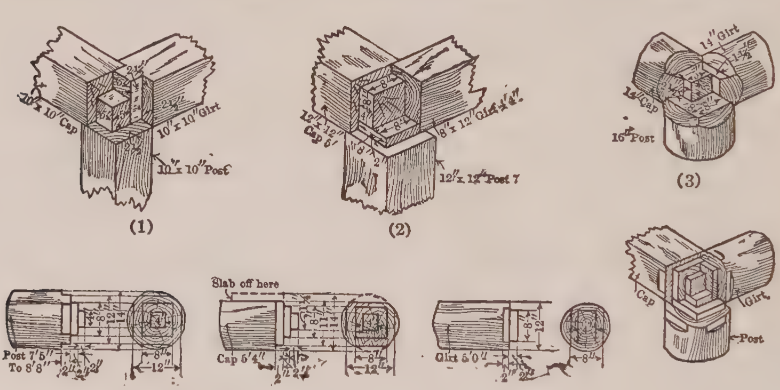

Fig. 176.—Square-set joints. (*Trans. A.I.M.E.*)

The type of joint used is one of two general types: The cap-to-cap set or the post-to-post set. These are shown in Fig. 176, 2 and 1. In the cap-to-cap set the maximum compressibility of the composite structure is in a vertical direction while in the post-to-post set the maximum compressibility is parallel with the caps. If we follow the principle of having a structure possess the maximum compressibility in the line of the greatest pressure, then where the side pressure is greatest the post-to-post joint should be used and the cap-to-cap joint reserved for maximum top pressure. If the maximum resistance with minimum compression is to be used against the direction of greatest pressure, then the cap-to-cap joint would be used where side pressure is greatest and the post-to-post where top pressure is greatest. Opinion is divided on this question. A wide variety of square-set joints has been described in mining literature. The step joint (Fig. 176) used by the Anaconda Copper Mining Co. has been found to be advantageous where round timbers are used. The joints shown are considered sufficient to represent prevailing practice.

The integrity of the square-set system of timbering depends upon the approximate alignment of the members of the system in the three planes. It is evident that the system admits a considerable degree of compressibility, and if top and side pressures were uniform over the area of a stope but little trouble would be experienced with the system. As this is seldom the case in practice, reinforcement is necessary where the members of the frame begin to get out of alignment. Diagonal braces, reinforcing sets, open cribs, and solid cribs are used and must be promptly applied where timbers begin to lose their alignment. Waste-filled stopes where the filling is conscientiously kept up close to the faces are more easily controlled in heavy ground, and it is seldom that a stope is lost even though timbers through irregular settlement may be 1 or 2 ft. out of line. Various methods of square-set stoping are given in the chapter on Underground Methods.

Lagging.—In order to prevent small pieces of rock from falling between the timber sets or main supporting members and also to concentrate the load upon these members, lagging boards are used. These boards may be 2-, 3-, or 4-in. planks of sufficient length to bridge the space between the sets. Usually planks a foot in width are used. Round poles, either whole or split, light steel rails, ribbed sheet metal, and reinforced-concrete slabs are used in place of planks. Top lagging, side lagging, close lagging, and spaced lagging are descriptive terms used; their meaning is apparent. "Lacing" is a miner's term used to describe spaced lagging used to confine filling in square sets or other similar situations. "Spiling" and "forepoles" are synonymous terms indicating lagging which is pointed or chisel-ended and is intended to be driven into position. In coal mines considerable use is made of flat stones in conjunction with pole lagging. The stones bridge the space between the lagging and are held in place by filling which is placed back of them. An economy of material results.

The first indication of pressure is seen in the bending of the lagging boards. The degree of deflection indicates the intensity of the load. Where loads are cumulative the first member of the supporting structure to yield should be the lagging boards. These should be weak enough to break before a sufficient load is thrown upon the sets to break them. Where very heavy ground pressures must be withstood, as in swelling ground, pegs may be advantageously used, as shown in Fig. 177. The pegs are driven into the lagging boards by the ground pressure and prevent loading up to the point of breakage. They are replaced when the lagging board closes down upon the set. In this manner lagging boards can be prevented from breaking, and thus excessive loss of timber in heavy ground is avoided. Still another method of preventing the failure of lagging boards and sets is to ease the ground back of them by picking it away. For this purpose spaces are left between the lagging boards, and when they begin to show excessive pressure the ground is dug out sufficiently to relieve the pressure.

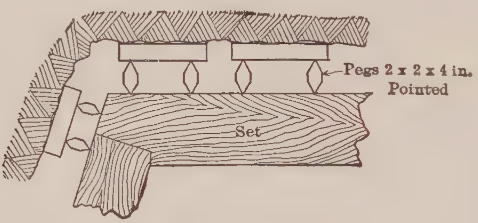

Fig. 177.

While timber lagging is generally used, various substitutes have been proposed and adopted to some extent. F. Drake describes and compares corrugated steel, buckled plates, and flat steel plates stiffened with angles. Flat steel plates with stiffening angles are considered too expensive as compared with other materials. Corrugated steel, 1/16 in. thick, is capable of carrying 200 lb. per sq. ft., while 2-in. plank will carry 360 lb. per sq. ft. (evidently plank with unsupported length of about 4 ft.). A buckled plate 1/4 in. thick would sustain 560 lb. per sq. ft., one 3/16 in. thick, 400 lb. For a strength equivalent to that of 2-in. lagging, two thicknesses of corrugated steel would be required. The comparative costs for wood and corrugated steel lining are given for a mine shaft. being, respectively, $2.12 and $5.94 per foot of shaft (area of lagging per foot is 55.5 sq. ft.)[1]

The principal advantage of metal lagging is that it is fireproof. Lagging constructed of reinforced concrete would accomplish the same end. Ferroinclave would be an excellent material for reinforced-concrete lagging.

Lagging is used in shafts, tunnels, drifts, and stopes and in fact wherever there is danger of small pieces flaking off without warning.

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[1] Use of Steel in Lining Mine Shafts. *Trans. Lake Superior Min. Inst.*, vol. 8, p. 34.

Braces, Sprags, Wedges, etc.—Braces or sprags are required in almost all kinds of timbering in order to provide against side thrusts and to steady important members until the weight holds them in place. Timbers of relatively small section—2 by 4 in., 2 by 6, 4 by 4, and 4 by 6 in.—are required. Tunnel sets are usually spragged at top, bottom, and middle. In square-set timbering, the girt member corresponds to a sprag. Drift sets in top slicing are spragged at three or four points.

Wedges are used in all timbering operations. They are used principally to hold the prop, cap, stull, or other member firmly until the ground pressure develops sufficiently to hold the timber in place. They also act as a cushion in distributing the load on a timber. One of their important uses is to fill in irregularities between the timber and the wall rock.

Bulkheads are placed in shafts and inclines that are being used for working purposes while in process of extension. They consist of heavy timber platforms from 1 to 4 ft. thick and heavily braced from below. Bulkheads are also used in stoping in order to confine filling and where hydraulic filling is used. In this capacity they act as retaining walls. The following empirical formulas[1] give the diameter and spacing of props used in the construction of bulkheads for the last-named purpose.

$$\left.\begin{array}{ll} \text{Flat workings} & D = L \\ \text{Chute workings (10–25 deg.)} & D = \tfrac{3}{2}L \\ \text{Pitch workings (25 deg. and up)} & D = \tfrac{7}{4}\text{L} \end{array}\right\} \begin{array}{l} L = \text{length of prop, ft.} \\ \\ D = \text{diameter, in.} \end{array}$$

Flat workings $S = W/H$ S = spacing of props center to center, ft.

Chute workings $S = \tfrac{2}{3}\dfrac{W}{H}$ W = width of opening, ft.

Pitch workings $S = \tfrac{4}{7}\dfrac{W}{H}$ H = height of opening, ft.

The props are placed in hitches at both ends, and the general depth of the hitch is equal to the diameter of the prop. The props are angled above the normal at an angle equal to one-seventh the dip of the coal seam. The construction of different types of bulkheads used in anthracite coal mining is shown in Fig. 178.[2]

Filling.—"Stowing," "waste filling," and "hydraulic mine filling" are terms used where mine openings are filled with rock, sand, tailing, gravel, ashes, or other material for the purpose of preventing subsidence and for support. Where working chambers are on an angle exceeding 45 deg. almost any kind of loose-rock filling can be used and deposited in place economically. With flat workings or workings at a low angle of inclination the placing of the filling involves a great deal of labor and

[1] *Bull.* 60, *U.S. Geol. Survey*, p. 43.

[2] *Bull.* 60, *U.S. Bur. Mines*, pp. 42–47.

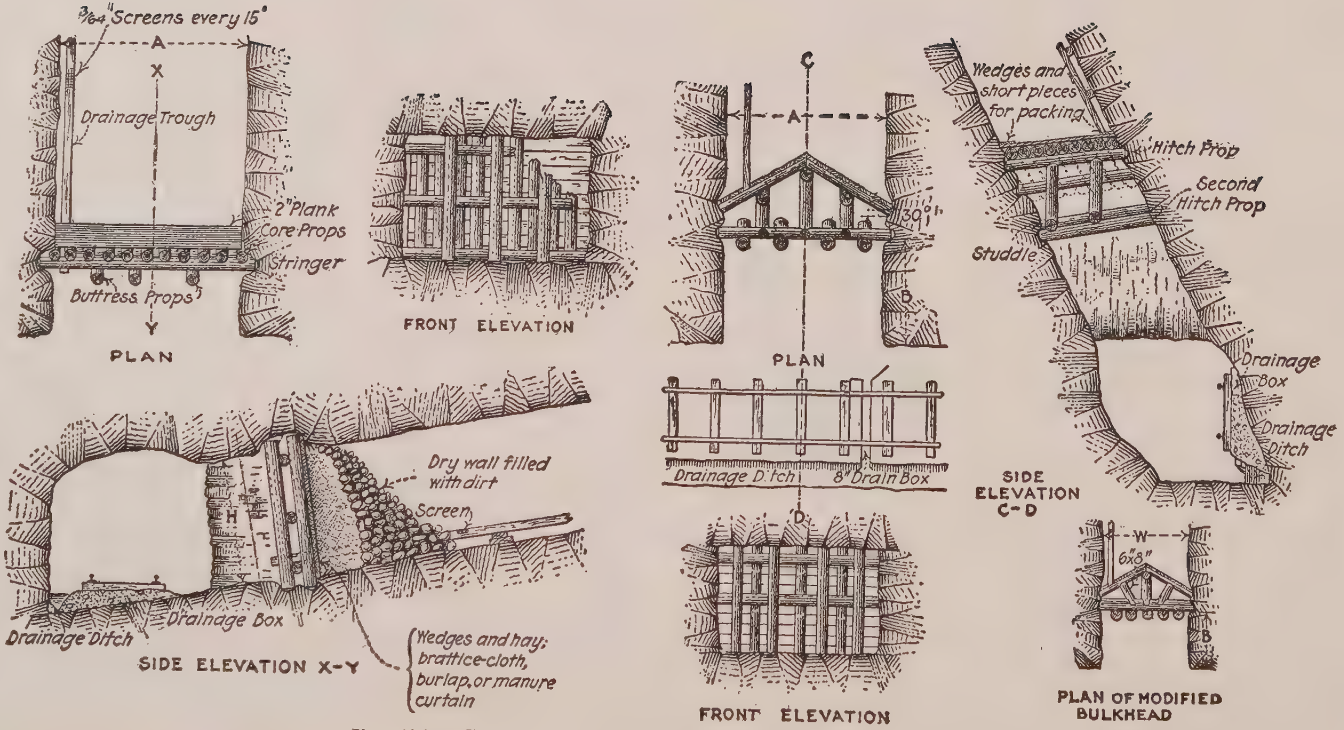


FIG. 178.—Construction of mine bulkheads. (*U.S. Bur. Mines.*)

cost. Transporting by water or hydraulic mine filling is the most economical method of filling flat workings and has largely displaced the cruder methods of hand stowing. To be most effective, the material must consist of pieces not coarser than ¾-in. diameter and must be mixed with finer constituents of such a nature as to form a cementing bond when deposited by the water. Culm, ashes, breaker refuse, sand, gravel, clay, and loam have been used. A mixture consisting of medium-sized gravel and a moderate proportion of loam has been found to be an excellent filler. Gravel mixed with sand, clay, and granulated slag serves the same purpose. The cost of obtaining the hydraulic filling material, together with cost of transportation, largely determines the selection of the material. Where rock is used it must be crushed to ¾-in. size or smaller.

Loose-rock filling shrinks to a greater or less extent after being placed in position. The amount of shrinkage in a given time is determined by the nature of the material and the pressure or load coming upon it. Filling material deposited by water is much more compact and shrinks less than loose-rock fills. The extent to which filling will shrink under variously applied loads and conditions is given in the following table:

Table 72[a]

Material confined and not free to expand laterally	Net tons per sq. ft. required to produce compression of per cent					Compression and load at end of test	
	3	5	10	20	30	Comp.	Load
Broken sandstone.......	3.33	5.55	13.32	46.6	98.6	35.0	666
Broken sandstone and sand................	3.5	5.77	24.42	308.5	......	23.0	686
Dry coal ashes.........	1.0	1.86	5.32	10.8	25.0	51.0	666
Coal ashes flushed in with water...........	......	......	......	5.50	22.0	51.0	666
Wet culm flushed in cylinder and partly dry..	8.9	14.28	35.52	138.7	444.0	33.0	668
Dry sand in cylinder....	3.0	5.27	33.3	129.0	499.0	32.2	666
Wet sand flushed in and partly dry...........	39.3	67.0	173.8	555.4	......	20.75	666

[a] *Bull.* 25, *U.S. Bur. Mines*, p. 55.

It is obvious that the loads applied at the end of the tests quoted above would be extreme and consequently the degree of compression would practically never approach these figures (assuming a weight of 150 lb. per cu. ft., a load of 666 tons per sq. ft. would represent a column of material 8,878 ft. in height).

Filling when it is first placed in position is under a pressure due to its weight alone and as a consequence the shrinkage is small. As time goes on it is subjected to a constantly increasing external pressure and the shrinkage becomes greater.

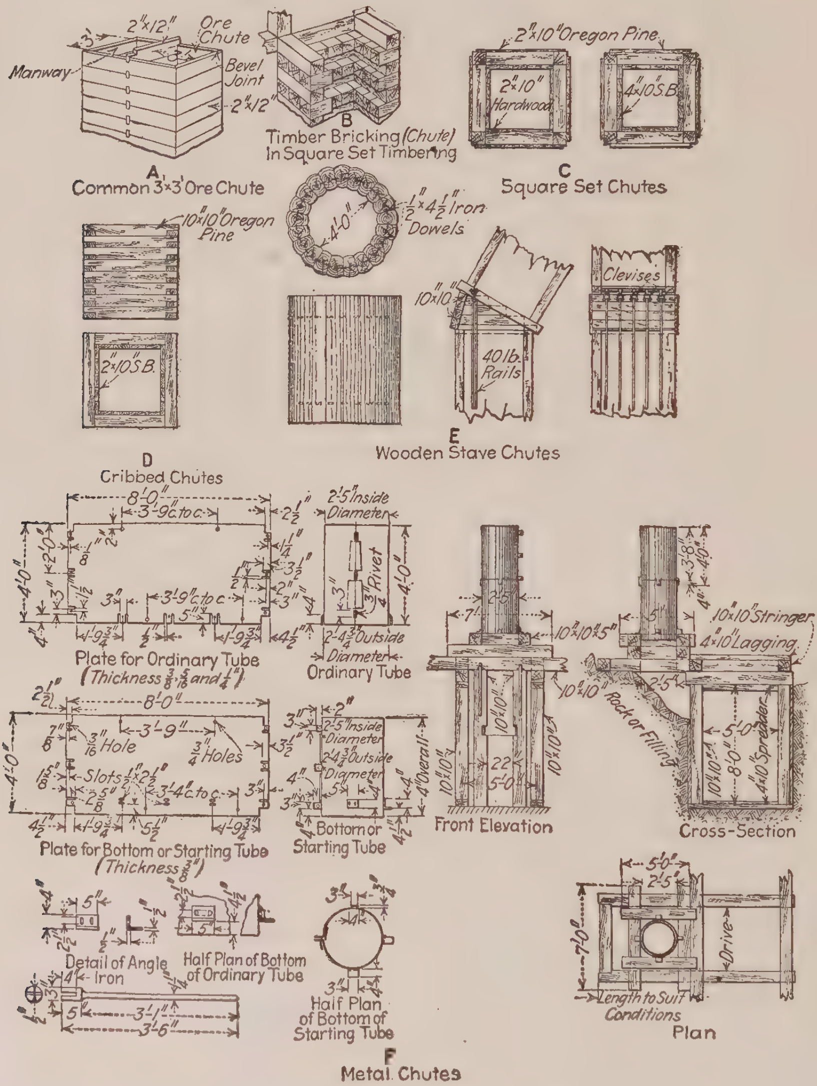


FIG. 179.—Construction of ore chutes.

Ore Chutes.—Ore chutes are constructed through waste fills. They require support to preserve integrity and to provide for the wear due to the passage of the ore or waste. Unlined passages through wall rocks or ore body are constructed where they can be suitably used. In square-set stoping, chutes are constructed by boarding a vertical line of sets with

2- or 3-in. planks. The planks are placed in vertical position and are replaced when worn out. Where the ore strikes against them in falling, steel plates or rails are placed in order to take up the impact. Where a chute is to be used a long time, timber bricking is used for the lining. The "bricks" are blocks 10 by 10 in., 12 by 12 in., or 12 by 14 in. They are placed with the grain of the wood normal to the vertical and a row of blocks is alternated with a stringer piece. The blocks project 2 in. beyond the face of the stringer. (See Fig. 179*b*.) The arrangements of a two-compartment timbered raise with ore chute, timber compartment, and ladderway are shown in Fig. 180.

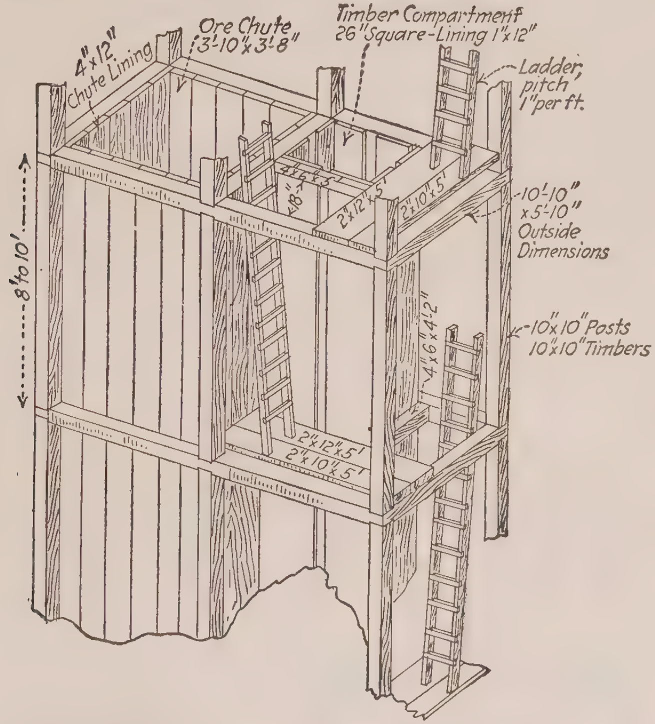


FIG. 180.—Chute and manway construction. (*Eng. Min. Jour.*)

Chutes are also constructed of close timber cribbing, the round timber being slabbed so as to leave a smooth interior to the chute. Chutes are constructed from 3 to 4 ft. square.

Circular chutes of 4-ft. internal diameter and supported by dry walls are used in some of the Michigan copper mines. Wooden and metal pipes surrounded by waste are also used for the same purpose. Several different methods of construction are shown in Fig. 179.

Jacket Sets.—In cases where "ground is on the move" or where ground pressures are cumulative, outer sets that enclose the main sets with a space interval between are used with shaft, tunnel, drift, station, or pump-chamber sets. By their use repairs to lagging and easing of the ground can be effected without disturbing the inner set. In Butte practice the jacket set is offset from the shaft timbers a distance of 2 ft. in the clear, making a total widening of over 3 ft., or a distance of more than 6 ft. in both width and length of the shaft section. Figure 181 is a plan of a Butte jacket set. Figure 182 shows another type of design,

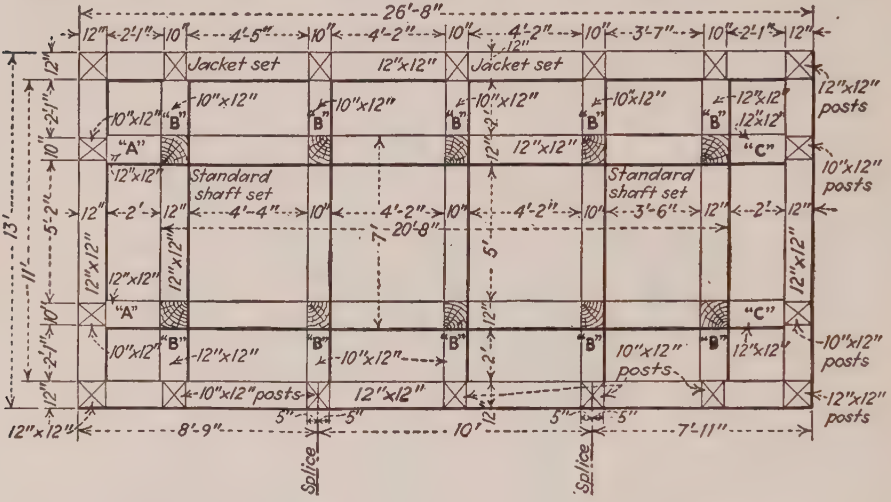


FIG. 181.—Jacket sets used in shaft timbering. (*H. R. Drullard.*)

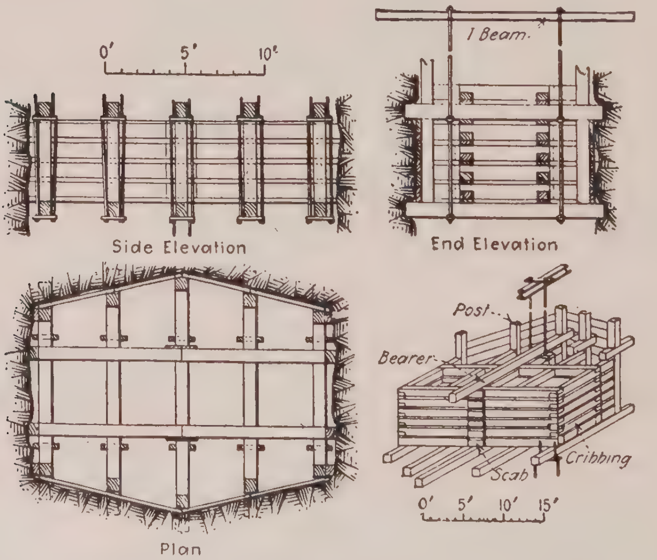


FIG. 182.—Jacket sets used at Shattuck-Denn mine. (*Joseph Walker.*)

in which the wall plates of the shaft are protected. A series of posts is carried upon bearing timbers and supporting horizontal lagging boards that form, in effect, a truss. The shaft timbers consist of cribbing supported by the bearers, as shown in the sketch. In other forms of this kind the jacket set is, in effect, a V truss, or a three-segment frame bearing at its ends upon the ends of the end plates.

Rooms and Stopes.—In the room-and-pillar system of mining coal the rooms are separated by pillars. They may be from 12 to 40 ft. in width and up to 300 ft. in length. The main weight of the roof is supported by the pillars, and as a consequence the props serve the function of preventing slabs and pieces of the roof from falling. Usually two or four or more lines of props are used parallel with the length of the room. The distance between the lines of props varies from 4 to 6 ft. Round-timber props are used, the diameter depending upon the length of the props and the pressure that must be supported. An approximate relationship is 1 in. of diameter to each foot of length. Headboards are used where the roof tends to slab in small pieces; where the roof is very bad, bars and lagging boards are used. The lines of props are placed as close as possible to the advancing face.

Four methods in the main cover the various mining practices for timbering rooms in top slicing. The first is the prop and headboard, used where the slice is not completely mined and where a thin layer of mineral is left in the roof of the room to protect the miners from any break-through of the loose material above. Drift sets with lagging boards are used where it is desired to mine all the mineral. Where a fresh advance is to be made, the excavation is started at the top and the lagging boards are driven ahead and caught on the edge of the top of the ore beyond. As this is done under the timber mat above, the miners are sufficiently protected. The excavation thus started is carried down under the protection of the lagging and the next drift set put in place and braced. The face is then ready for another advance. The maximum height of slice taken in this manner ranges from 10 to 15 ft. Square sets are used in top slicing where the thickness of the slice is irregular, as is usually the case with the first or uppermost slice. Rooms are sometimes carried up to four sets high and of irregular width and height. Under an unstable roof the driving of lagging boards is resorted to. Usually relatively light members are used for the square-set members. The "stringer-and-prop" method is used for slices not to exceed 10 ft. in thickness. In this method stringers or sills are laid alongside of the props, and boards (2 in. thick) are laid over them, forming a tight floor. When the next slice is taken out the props are placed under the stringers above at intervals of about 5-ft. centers. The miners work under the protection of the floor of the slice above.

In working stopes in narrow veins a variety of methods are available. They are summarized below:

1. By the use of pillars equal in thickness or diameter to the width of the stope (minimum dimension).
2. By vertical ribs at either end of a stope and longitudinal pillars at the bottom and top of the block.
3. By bottom, top, and end pillars, and ribs or pillars between.
4. By stulls.
5. By batteries of stulls, or spliced stulls.
6. By stulls and stringers on the hanging wall.
7. By stulls and stringers on both the hanging wall and the footwall.
8. By stulls, stringers, and lagging on the hanging wall.
9. By stulls, stringers and lagging, and plates or stringers on the footwall.

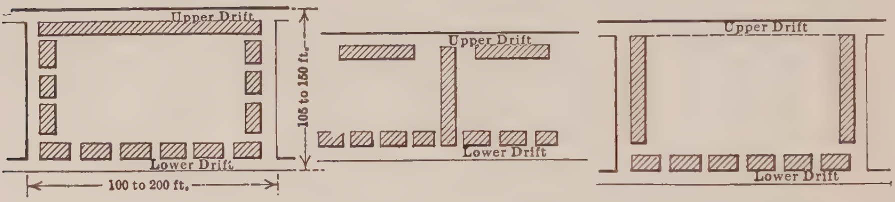


Fig. 183.—Pillars used in supporting stopes.

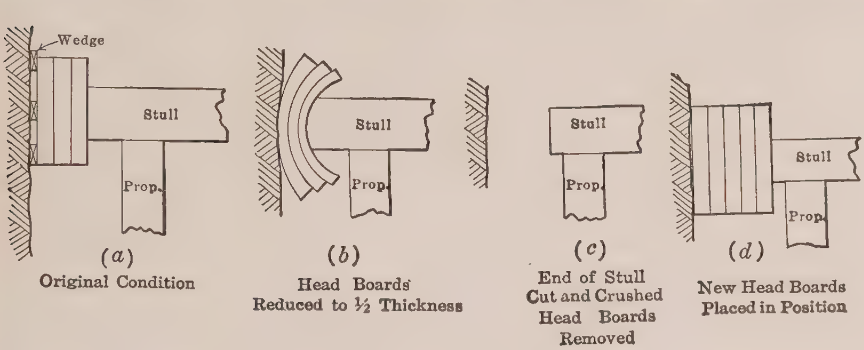
10. By stulls and partial filling. Several rows of stulls may be planked and filling placed within.
11. By leaving pillars and afterward filling by sand flushing.
12. By cribs where the working is comparatively flat.

The first three methods involve the use of pillars, and these are justified where the ore is of very low grade or where portions of the vein are barren and such barren portions are readily distinguishable from the ore. The arrangement of pillars admits of wide variation, ranging from occasional pillars of irregular dimensions to pillars systematically placed. Figure 183 shows the possibilities of systematic pillaring.

Methods 4 to 9 inclusive involve the use of the stull. Stulls are placed systematically. Where stulls of insufficient diameter must be used, groups of three are placed as a battery and bound tight to obtain sufficient strength. The limit of width for stulls may be taken as from 12 to 20 ft. Stringers or long timbers interposed between the stull and the walls are required where lagging is needed. Vertical props or posts are used frequently to brace long stulls. The system bears a close resemblance to the square-set method.

Stulls may be set in hitches in both the hanging wall and the footwall, or they may be placed in a hitch on the footwall with a headboard and wedges used on the hanging-wall end. The latter is the method more often found in practice. Where stulls are used on inclined veins they are set at an angle of 10 deg. above the normal between the walls or at right angles to the wall. The settling down of the hanging walls tends to bring the stull into a position practically coinciding with the normal.

Where the ore body is vertical, horizontal stulls are used and these permit the use of headboards on either end. They are wedged into place, and vertical props keep the stulls in place until the side pressure develops sufficiently to hold them firmly. The further development of side pressure crushes the headboards without breaking the stulls. When the headboards are crushed down to 40 or 50 per cent of their original thickness the ends of the stulls are sawed off and the new headboards are placed. In this case a stull that would take weight enough to crush within a month's time without headboards will last some 12 months or longer. The method is shown by the four sketches in Fig. 184.



(*a*) Original Condition

(*b*) Head Boards Reduced to ½ Thickness

(*c*) End of Stull Cut and Crushed Head Boards Removed

(*d*) New Head Boards Placed in Position

Fig. 184.—Compression and replacement of headboards in stull timbering.

Methods 10, 11, and 12 involve the use of filling as a final means of support, whatever timbers are used for the purpose of temporary support. In the case of an ore body of considerable vertical and lateral extent, open-timbered stopes are inadvisable and filling must be resorted to. While it is true under such conditions an open stope of considerable size can be maintained, the larger the stope, the greater are the forces which tend to produce movement, and as a consequence when movement starts it becomes much more difficult to control. When development shows an extensive ore body, the sooner the stopes are filled and the more systematically filling follows the ore excavation, the simpler is the support problem.

Wide Ore Bodies.—Where walls and ore are firm and solid, chambers of large size may be excavated without support of any kind. The limit to size is uncertain and admits of no well-defined rules. Filling of some kind is invariably resorted to unless a top-slicing method or underhand stoping is used. The mining of a slice of ore is followed by filling, the filling serving the double function of giving support and affording a working platform for the miners. The filling is more frequently placed in horizontal layers, although sometimes it is run in on a 45-deg. slope and follows up the advancing ore face, which is maintained at approximately the same slope. The maximum length of stope is determined by local conditions and by the judgment of the engineer. Narrow vertical ribs or pillars separate the stopes along the strike. Two hundred feet may be taken as the maximum length of the stope.

Where the ore back is not firm enough to support itself across the width of the ore body, cribs can be constructed on the waste fill at frequent intervals to support the weight. The timber cribs may be buried in the fill or else recovered in part.

In some ore bodies the broken ore itself may be left in the stope to serve as a support and a working platform. This method has been described under shrinkage stoping in the chapter dealing with Underground Methods. If walls and ore body are weak, square set and fill is the only method suitable if block-caving is out of the question.

Trenches and Foundations.—Trench and foundation excavations require little or no support where they are excavated in relatively firm materials or where the sides of the excavation can be sloped. Excavations driven in sand-like material or where the vertical walls have to be carried in materials tending to cave require temporary support. The simplest form of trench support is the sheet pile. This is a 2- by 6-in. sheet plank from 8 to 12 ft. long. One end is chisel-shaped. Two rows of piling are driven along the lines of the trench and the earth between is excavated and removed. Longitudinal stringers and cross braces are put in as excavation is deepened and serve to stiffen the sheet-piling wall. Deeper excavations are supported by driving a second double row within the first and extending this down as in the upper row. An excavation of considerable area can be driven in the same manner, the only novelty being the necessity of extending the braces from wall to wall in both directions. Figure 185*a* gives the details of a trench excavation. Wooden sheet piling is driven with a heavy wooden maul, the head of the pile being protected by a driving cap. Compressed-air hammers have displaced hand methods for driving and withdrawing sheet piles. The piles are pulled after backfilling by a pulling yoke, shown in the figure. A single row of sheet piling serves for dry loose ground. With wet ground a double line of piles is better, and more satisfactory still is a double line of grooved piles, in the grooves of which a tongue is driven.

For deep trenches in wet soft ground the steel sheet pile has displaced the wooden pile and is much more satisfactory. A number of kinds of interlocking steel sheet piling are on the market. The U.S. Steel and Lackawanna sheet piling are shown in Fig. 185*b* and *c*. The Lackawanna is made in the straight-web, arched-web, and center-flange types, the constructions other than the web type giving a much stiffer pile. Steel piling is practically watertight and hence is of great service is wet foundation work in soft ground. It can also be used in shaft work, and for this purpose piles up to a length of 85 ft. are available. It is necessary to use interbracing in order to prevent the walls from buckling as the ground is removed from within the enclosure.

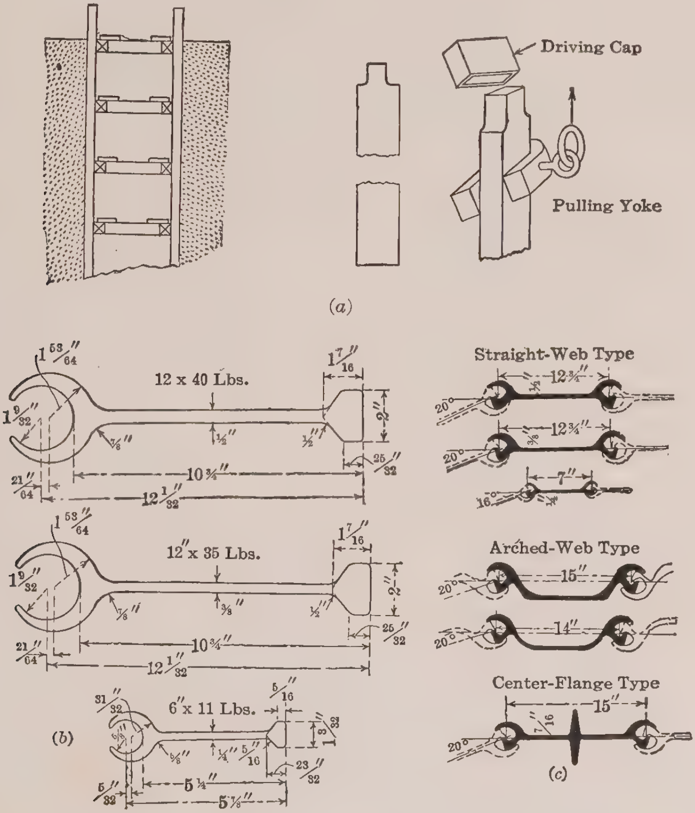


Fig. 185.—Wooden and steel sheet piling.

Retaining Walls.—Retaining walls are frequently used in and about surface plants. Very often the filled timber crib is employed. Wooden walls are anchored by wire rope attached to deadmen. The subject of the mechanics of retaining walls is comprehensively treated in civil engineers' handbooks, and to these the student is referred for extended discussion of this structure.

Handling Timbers.—Mine timbers of ordinary lengths and dimensions such as would be required in square-set stoping, drifting, and narrow stopes are lowered into the mine on cages, or skips. Where cages are provided with doors and the decks are fully loaded no lashing is necessary, but without doors or with partially loaded decks the timber load must be lashed. With skips the timbers are loaded without difficulty. Where large timber cages are in use the timber is loaded into a car and run on to the cage. This is the most economical method of handling timber and involves the least delay in loading and unloading cages. Long stulls can be loaded only on the top deck of two- or three-deck cages. In some cases two-deck cages of special construction are required. In them the upper deck is made removable so that the long timbers can then be loaded on the lower deck. Shaft timbers are lowered by attachment to a chain or rope fastened beneath the deck of the sinking cage. They are swung into position and attached to the suspension rods before being released.

Unloading timber at stations is done by hand. Chain blocks and crabs are used for very heavy timbers, and the labor required in unloading is reduced to a minimum.

Wedges, blocks, and miscellaneous small timber are loaded into cars, lowered to the stations, and trammed to the points where required. For convenience in handling, blocks may be nailed to a strip so that a number can be handled as if there were one piece. Wedges are handled in bundles.

Transportation from station to stope is effected in special timber cars, which are low flatcars provided with iron side supports where round timbers are handled and without them where square timbers are in use.

Getting the timbers into the stopes is effected by either hoisting them up or lowering them down through a manway. In some cases an inclined timber chute with a deflecting board at the floor on which the timbers are to be discharged is used and the timbers dropped into place. In the Minnesota iron mines, timber chutes leading from the surface to the sublevels are used, and thus all lashing and unlashing of timbers is avoided. In the great majority of cases timbers are placed in position manually. For the placement of very heavy stulls, temporary props are placed in position and a block and tackle is used to hoist the timber into position. Chain blocks of light construction also serve the same purpose. A temporary rigging is required in either case. Light jacks of considerable extensibility are also used to lift heavy timbers.

Miscellaneous.—Prop and post pullers are used in recovering mine timber in coal mining. They are necessary because of sudden falls of roof after the removal of the prop. The Sylat post puller is shown in Fig. 186. Timber is seldom recovered in metal-mining operations, although certain economies can be effected by systematic efforts in this direction. C. L. Larson described the methods used in square-set mining.[1] The method consists in taking a section of the open stope, three sets wide and six sets high, lacing the sides of the section, and catching up the timbers forming the bottom of the second floor by diagonal props or stulls. The same thing is done for the third floor and then the posts, caps, and girts of the sill and first floors that are within the area are removed and the space is filled from a chute served at the top of the stope. The lowest line of stulls is removed and replaced on the fourth floor and the timbers constituting the second floor are removed, with the exception of the outer members, and then more filling introduced. Floor after floor is treated in this manner until the top has been reached. The top floor can be "handbanked" if there is a bad back, or many of the timbers of the topmost floor can be recovered by careful work and the use of props resting on the filling. Where excessive side pressures from neighboring filled stopes or ore faces develop during the timber recovery, some of the inclined stulls can be left or a horizontal stull spanning the width of the filled stope can be placed. Figure 187 shows the method. In the figure a line of sets is left open for attacking the ore face from the side. Larson shows that work of this kind is profitable.

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[1] *Min. Eng. World*, May 24, 1913, vol. 38, p. 985.

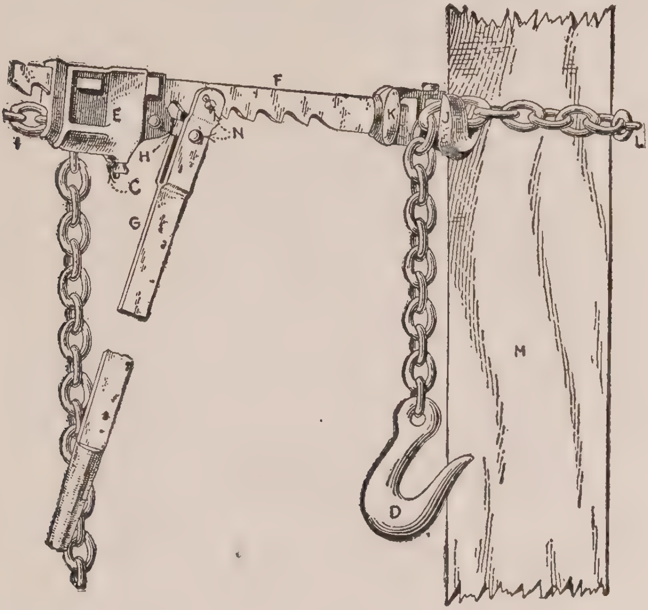

FIG. 186.—Sylat prop puller.

Fundamental Principles.—The following summarizes the more important fundamental principles applying to mine support.

Other things being equal, where timber is used the timber that is toughest, strongest, and longest lived should be selected in preference to cheap timber.

Where ground pressures are heavy, uncertain, or cumulative, soft wood should be used in preference to hard woods, malleable metals in preference to nonmalleable, and masonry and timber composite structures in preference to all-masonry structures.

As far as practicable, timber or other materials used in support should be subjected to compressive forces.

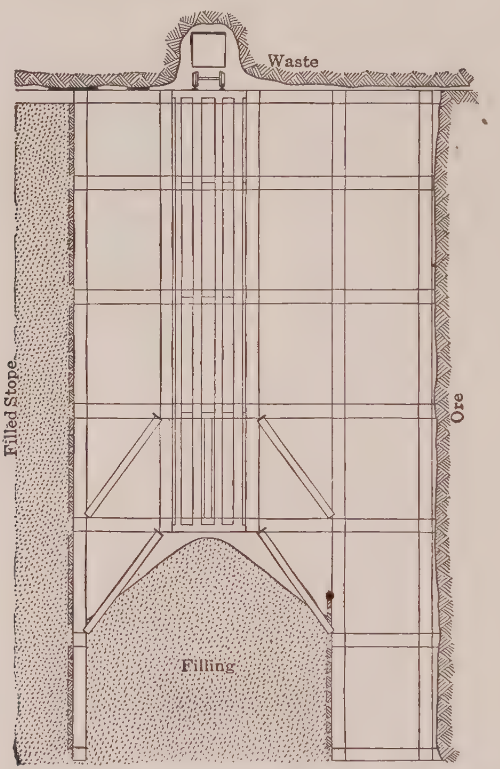

Fig. 187.—Timber recovery in square-set stoping.

Joints should be of the simplest construction and should be designed for compression, the minimum weakening of the timber, the minimum loss of material in framing, and the minimum risk of breakage in handling.

Timber members should be standardized and their preparation by machines instead of by hand labor should be provided for.

The design of the sets should be such as to equalize the loading and to balance the thrusts. No dependence should be placed upon bolts, spikes, or rods.

The design and materials selected should admit of yielding either by condensation of the structure or by penetration of part of the structure into the ground where heavy ground or cumulative ground pressures must be resisted.

In heavy ground or cumulative pressures, provision for easing the ground should be made.

No large open spaces should be left back of lagging or main supports. Such spaces should be filled with timber, timber and waste, or, if economy is necessary, by cribbing with refuse timber. Whatever filling is used should possess the property of consolidating to a greater or less extent as pressure comes upon it. Its immediate purpose is to prevent the fall of a heavy block of ground upon the lagging or timber sets.

Timber treated with preservatives is justifiable where long life is a desideratum.

Comparative first cost and maintenance should be figured for each material and the selection made on the basis of lowest unit cost. The unit cost, which involves the element of time, might be expressed as cost per cubic foot of excavation per year of life in working openings.

In stoping heavy ground the least possible volume should be opened out as an unsupported space and then timbered as quickly as possible. Where the ground is loose or especially liable to cave without warning, forepoling of some kind is essential.

Where block movement is expected in the mining of large ore bodies, filling is necessary in all overhand stoping.

The mining of large ore bodies where top slicing or block-caving is impossible and where the ore body requires support is best accomplished by timbering supplemented by filling.

Large ore bodies where the ore is "heavy" are best divided into small blocks, and each block should be mined and filled as a unit before the blocks immediately adjoining are touched.

Timbering or support is a means to an end and only sufficient money to accomplish the specific end should be put into the supporting structure.

The subject of support in a given mine should be studied and deductions made as the mine is opened out. The experience gained by a certain amount of experimentation with materials and methods should be made a matter of record and used in planning new work. The experience of neighboring mines is often of great value to the management of a new mine.

Gunite.—The application of cement-sand mixtures, conveyed dry by a stream of air under high velocity and deposited by a mixed water and air spray upon the surface to be coated, has received considerable acceptance in mining work. The layer deposited in this way is exceedingly dense, practically water-impervious, and usually bonds well with the surface upon which it is deposited. It can be used to prevent slabbing where slabbing or flaking off of the rock surface in mine workings is caused by the action of air and moisture upon the exposed rock surface. The layer of gunite, being impervious to both air and moisture, prevents the slacking or slabbing of the rock. By its use timbering can be avoided under the conditions described. As applied to timbers, it is an effective fireproofing agent. Experience has shown, however, that when important mine timbers are covered with a gunite layer they cannot readily be inspected and their condition determined without breaking the layer. Dry rot takes place underneath, and for this reason gunite over timbers is of doubtful advantage.

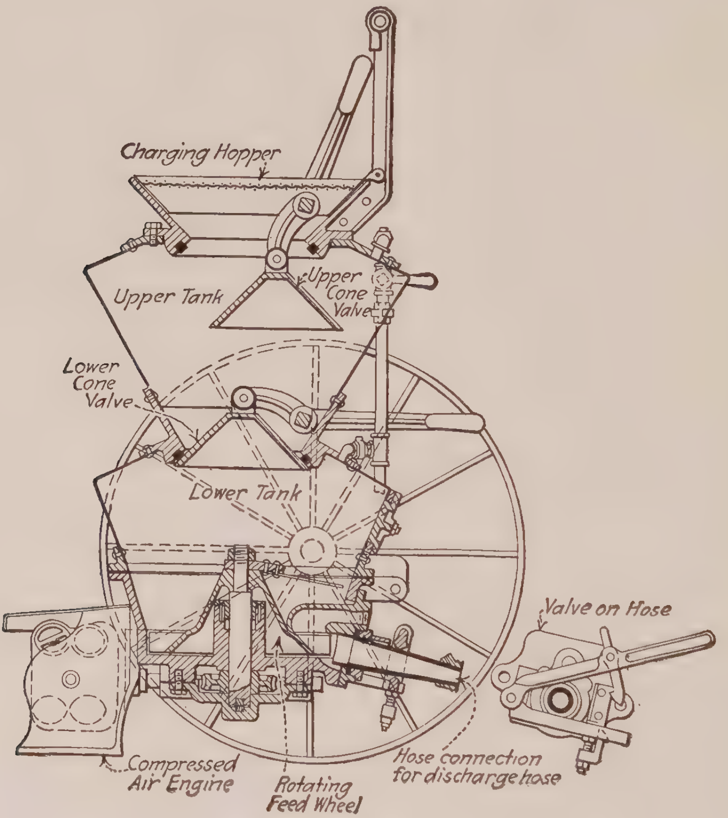

Fig. 188.—Sectional view of cement gun.

Clean dry sand (maximum size limited to ¼ in.) is used in proportions ranging from 1 to 3 up to 1 to 5. For coatings ¾ in. thick or less, E. M. Norris recommends a mixture of 1 cement to 3 of sand. George S. Rice states that due to rebound material a mixture of 1 to 3 approximates 1 to 2½ when in place. For a 1 to 4½ mixture, one bag of cement (1 cu. ft.) is stated to be sufficient for 32 sq. ft. of gunite 1 in. in thickness; for a 1 to 3 mixture the area is 22 sq. ft. 1 in. in thickness. Approximately 1 cu. yd. of mixture will equal 0.6 cu. yd. of gunite in place. Three sizes of machines are made for handling the mixture. These are designated N-0, N-1, and N-2 and have cubic contents, respectively, of 3, 4, and 5 cu. ft. Their weights are 900, 1,400, and 1,475 lb. complete with standard equipment. Sizes of the material, air hose, and water hose are, respectively, 1 in., ¾ in., and ½ in.; 1¼ in., 1¼ in., and ½ in.; 1⅜ in., 1¼ in., and ½ in. Free air required is 110, 150, and 225 cu. ft., respectively. Water pressure should be 20 lb. in excess of air pressure, which ranges from 40 to 75 lb. Maximum length of material hose is stated to be 500 ft., although 200 ft. is probably nearer the practical limiting distance. E. M. Norris found that in shaft work with 75 lb. air, the cement gun worked well at a distance 200 ft. above and 75 ft. below the gun. Figure 188 shows a cross section of the cement gun.

Except where it is used for waterproofing, there is no special advantage in using gunite for floor work. For wall work there is a limit to the thickness of the layer that can be applied, as thick layers tend to slough off before the cement sets. In work of this kind several successive layers are applied until the desired thickness is attained. In roof work there is a still greater tendency to slough and the layers should be thinner, approximating ¼ in. or less in thickness for each layer. Chicken-wire reinforcement is necessary in covering timber. This is first attached to the timbers and then the gunite is applied.

Quantity of Timber Required.—The quantity of timber required per ton has been figured out in the following examples.

TABLE 73.—TIMBER QUANTITIES AND RATIOS FOR A THREE-COMPARTMENT SHAFT—COMPARTMENT SIZES: 4.5 BY 5, 4.5 BY 5, AND 6 BY 5; EXCAVATION LINE 1 FT. OUTSIDE OF TIMBERS

Size of timber	Board feet				Volume excavat.	Bd. ft. per cu. ft.	Bd. ft. per ton
	Shaft set	Blocking and wedges	Lagging	Total			
10 × 10	855	200	600	1655	1066	1.55	20.2
12 × 12	1256	250	600	2106	1080	1.95	25.4
14 × 14	1370	300	600	2270	1188	1.91	24.8

TABLE 74.[a]—BOARD FEET PER LINEAR FOOT OF DRIFT

Size of timber, inches	Spacing of sets center to center			
	6 ft.	4 ft.	3 ft.	2 ft.
6 × 6	31.0	35.5	40.0	49.0
8 × 8	38.7	46.8	54.9	71.0
10 × 10	49.3	62.4	75.3	101.3
12 × 12	62.0	81.0	102.0	138.0

[a] 2-in. solid lagging on top; lagging 3 ft. wide on sides; drift 4 by 6.5 ft. inside timbers—no waste allowed.

TABLE 75.—BOARD FEET PER TON OF ORE REQUIRED BY STULL TIMBERING

Width of vein, ft.	Spacing 6 ft. both ways	Spacing 6 ft. hor., 7 ft. vert.	Spacing 6 ft. hor., 8 ft. vert.	Size of stull
6	0.85	0.72	0.70	6 in.
8	2.02	1.3	1.10	8 in.
10	2.3	2.0	1.7	10 in.
12	3.4	3.0	2.5	12 in.
16	3.4	2.8	2.5	12 in.
		Tons of ore per stull		
6	16.6	19.3	22.0	
8	22.1	25.8	29.5	
10	27.7	32.3	36.9	
12	33.2	38.7	44.3	
15	44.3	51.7	59.0	

NOTE.—Includes stull timbers only—no ladders, chutes, manways, or drift lagging. Sp. gr. ore, 2.67; 13 cu. ft. per ton.

TABLE 76.—BOARD FEET PER TON OF ORE IN SQUARE SETTING—SP. GR. ORE, 3.5; SP. GR. 2.67

Size of timbers	Square-set interval			
	5 × 5 × 7.5		6 × 6 × 8.5	
	Sp. gr. 2.67	Sp. gr. 3.5	Sp. gr. 2.67	Sp. gr. 3.5
8 × 8 in.	6.4	....	4.4	3.2
+ 20 per cent excess	7.7	....	5.3	3.8
10 × 10 in.	10.0	7.1	7.0	5.1
+ 20 per cent excess	12.0	8.5	8.4	6.1
12 × 12 in.	14.5	10.0	10.1	7.3
+ 20 per cent excess	17.4	12.0	12.1	8.8
Tons per block	14.5	20.5	24.3	33.5

TABLE 77.—BOARD FEET PER TON IN STOPING

Cut and fill	1–7
Shrinkage	1–4
Square setting	12–20
Top slicing	Variable to 12
Butte mines average	14

TABLE 78.—BOARD FEET PER TON—UNITED VERDE

(*J. B. Pullen*)

Remnant stopes	7.47
Vertical pillars	5.89
Horizontal pillars	9.26
Mitchell-slice stopes	7.18
Inclined cut and fill	4.08

TABLE 79.—BLOCK-CAVING—TIMBER, STEEL, AND CONCRETE PER TON
(*P. B. Bucky*)

Mine	Inspiration	Miami	Emma Nevada	Johnson
Timber, bd. ft.	1.00	1.58	1.75	0
Steel, lb.	Small	Small	Negligible	0.68–0.96
Concrete, cu. ft.	Small	Small	Negligible	0.054–0.03

TABLE 80.—CONCRETE FOR SHAFTS AND COST OF CONCRETING
(*M. J. Elsing*)

Shaft	Cu. yd. per ft.	Cost per ft.	Concrete shaft		Outside area, sq. ft.	Wall thickness, ft.
			Cost concrete per cu. yd.	Cost concrete per cu. ft. of shaft		
Junction	5.5	$ 79.00	$14.36	$0.27	288	1.7
Denn	4.5	80.25	17.82	0.38	210	1.7
United Verde	2.4	146.84	61.30	0.72	205	0.7
United Verde—ring	0.8	50.11	58.80	0.24	205	
Brier Hill	3.0	41.00	13.70	0.19	216	1.6
Negaunee	3.2	39.50	12.34	0.13	314	1.5

TABLE 81.—PROPS REQUIRED IN ROOM AND PILLAR, ILLINOIS COAL MINES
(*Illinois Coal Mining Investigations*)
Maximum number 8.1 props per 100 sq. ft.; minimum 0.9; average 5.0

TABLE 82.—SUPPORT COSTS PER FOOT AND PER TON
(*U. S. Bureau of Mines*)

Timber cost per foot		Timber cost per ton	
Drifts, crosscuts	$0.00– 4.37	Open stoping	$0.00 –0.10
Board ft., per ft.	0–137.8	Shrinkage stoping	0.006–0.20
Shafts, timbered	6.30– 13.15	Cut and fill	0.07 –0.47
Shafts, steel-concrete	45.90–111.45	Square setting	0.26 –0.40

Cost of Support.—The elements that enter into the cost of support are cost of material, preparation, handling and bringing to position, placing in position, replacements, easing of ground, and inspection. Development workings must be kept open, and while considerations of economy require the immediate abandonment of such workings when they have served their purpose, a relatively large proportion must be kept open and in repair. The aggregate cost of this may be charged to development or may be placed as an item in the cost of mining. The cost of support in actual stoping is readily figured on a per-ton-of-ore-mined basis and is an approximately constant quantity in any given ore body. Comparative costs can be roughly determined by comparing the amount of timber required or where different materials are to be used, by computing costs for the different types of structures.

Subsidence at Surface.—Subsidence research has produced a literature in several languages. Conclusions are diverse because so much depends upon the nature of the overlying rocks and the presence of shearing, slip and fault planes, stratification and dip, as well as the mass of mineral mined, its shape, position, and depth, and whether waste-rock filling has been used. The area of subsidence may be equal, greater than, or less than the area mined; the surface depth of the subsidence may be some fraction of the thickness of the mineral mined; subsidence may begin several months after mining is completed and may continue a number of years thereafter. Depth influences this factor, shallow deposits subsiding more rapidly than very deep ones.

One of the most complete studies of subsidence in metal mining was made by W. H. Kantner[1] for subsidence over caving areas in an ore mass at Bisbee. The original thickness of the capping was 425 ft.; the total height of the irregular ore mass was 625 ft. The upper horizon was 400 by 150 ft.; at the 900-ft. depth it was 700 by 500 ft.; and at the 1,060-ft. depth, 100 by 100 ft. Final measurements after 77 months showed 28,250,000 cu. ft. of ore mined; the projected area of caving blocks was some 161,000 sq. ft.; the surface area subsided was 1,535,000 sq. ft. with volume 15,855,600 cu. ft. and a maximum subsidence of 78 ft.; ratio of volume mined to volume of subsidence area was 1.78. The maximum angle of break was 45 deg. above horizontal; and the minimum, limited by a fault, was 65 deg.

Barodynamics.—The application of models in the study of the action of mine-supporting structures and roofs and walls in relation thereto promises a better understanding of the problem of mine support. Professor P. B. Bucky and his assistants have developed an ingenious centrifugal machine in which a model to scale of an original or prototype may be tested by forces that may be measured. Bucky states that a scale model of any weighty structure will behave similarly to its prototype if the model and the prototype material are the same and the model weight is increased in the same proportion as its linear dimensions are decreased. He obtains the effect of an increase in weight by substituting a centrifugal field of force for the gravitational one. *Technical Publication* 425, *A.I.M.E.*, 1931, contains a discussion by Bucky of his method, and subsequent papers give results of researches in this new field. The method is not without its limitations, and its conclusions may in some cases be open to considerable doubt, but there is no doubt about the value of such researches in aiding in the understanding of ground forces and supporting structures.

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[1] *T.P.* 552, *A.I.M.E.*

The U.S. Bureau of Mines has made interesting studies of roof movements in coal mines (*Report of Investigations* 3452, 3355) and has initated research in the measurement of pressures on rock pillars (*Report of Investigations* 3444). C. A. Pierce (*T. P.* 985, *A.I.M.E.*) gives the results of measurements of roof and floor convergence in mining a mixture of sylvanite and halite in horizontal beds varying from 5 to 12 ft. in thickness under a cover about 1,000 ft. in depth. The average weight of cover approximated 1,000 lb. per sq. in.; a 50 per cent extraction by room-and-pillar mining loads the pillars to 2,000 lb. per sq. in., which Pierce states is their approximate bearing strength, or compressive strength just short of producing plastic flow. Room-area extraction on first mining is about 58 per cent and total panel area extraction 40 per cent. Convergence was found to be of small magnitude. No conclusions were made and the studies are continuing.

Important studies of earth pressure and shearing resistance of plastic clay in tunnel and open-cut work were made by K. Terzaghi, R. B. Peck, and W. S. Housel and are given in a symposium report, *Trans. Am. Soc. Civil Eng.*, No. 108, 1943.

Rock Bursts.—Discussion of rock bursts is a live subject among mining engineers in relation to very deep mining. A rock burst is a sudden, violent breaking of rock walls stressed beyond their strength limits and is accompanied by earth tremors. In the Kolar gold field (India) and in the deep mines of South Africa, rock bursts caused 40 per cent and 9 per cent, respectively, of all fatal accidents for the years 1932–37, inclusive. The potential hazard of rock bursts is a serious factor in some but not all deep mining. On the other hand, rock bursts may occur both at the surface and at comparatively shallow depths. However, they are much more likely to occur in the relatively few mines whose depths exceed 3,000 ft. Gold mines in Ontario have reached depths of 4,000 to 6,000 ft. In some mines control of rock bursts has assumed great importance. In the India districts there are five contiguous mines along the same ore zone, two of which have reached depths of 7,000 ft. or more and all but one are below 6,000 ft. The Kirkland Lake area (Ontario) comprises a contiguous group of some six mines, two of which have reached depths of about 6,000 ft. In South Africa a number of contiguous deep mines operate along the same ore zone. In all three of these areas rock bursts are of great concern. Rock bursts likewise occur in Sudbury (Ontario). In the United States the Michigan copper district has had some rock bursts, but no rock bursts have been reported in California, where two contiguous gold mines have reached depths of 5,000 ft.

Partial control of rock bursts has been effected by reducing open stoping space to a minimum, systematic sequence in stoping, careful waste-rock filling, the use of stronger stope supports, and retreating methods. Pillars and unmined stope remnants are avoided, as there is a concentration of pressure upon all residual mining areas when stoping nears completion in any given place.

In the Kolar district, P. J. Crowle makes use of squared granite blocks to support stopes where filling by waste rock proved insufficiently rigid and concrete proved to be too rigid. Granite blocks allow a compression of some 10 per cent and are very strong. By the use of timber filler webs a somewhat greater degree of compression may be obtained with the granite blocks. Drifts are supported by 75-lb. steel rails bent to form two halves of an elliptical section when joined at top and bottom by ⅝-in. welded-plate joint boxes. The steel frames are placed at 4-ft. centers and support round-pole lagging, behind which is placed not less than 18 in. of loose packing.

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CHAPTER XII

OPEN-PIT MINING

GENERAL FEATURES

Advantages and Disadvantages.—Open-pit mining has the advantages of lower mining cost, easier supervision, larger mechanical appliances, better working conditions in respect to light and air, usually safer working, no timber for support, more convenient and usually more efficient use of explosives, larger production, fewer skilled workers, higher work ratios, and a greater proportion of mineral won over that obtained by underground mining. It has some distinct disadvantages, such as exposure of the workers to the weather and as a consequence the restriction of working in some localities to a part of the year, the limitation to moderate depths, a plant made up of a number of scattered units, and frequently the removal of large amounts of waste or overburden material.

Limitations.—Obviously the mineral deposits such as creek, bench, and river placers and those deposits that outcrop on the surface are best won by open-pit working, while deep-seated mineral deposits can be worked only by underground methods. Veins and beds that are inclined at high angles cannot be mined to special advantage by the open-pit method unless they are very thick. Extensive flat deposits buried under a moderately thick cover or directly exposed are most advantageously mined in this manner. Mineral deposits can be completely or partially won by open-pit methods. The varying thickness of overburden and the structure of the deposit determine this feature. The relative amounts of open-pit and deep ore within certain limits influence the selection of the method. A deposit comprising a small amount of open-pit ore and a large amount of deep ore would be mined by underground methods. On the other hand, it is comparatively easy to use underground methods from the edges of an open pit.

The bounding line between open-pit and underground mining for a given deposit is established by the cost differential between open-pit and underground mining and the cost of surface excavation.

CLASSIFICATION AND PHYSICAL PROPERTIES OF ROCKS

Classification.—In excavation work the engineer classifies rock masses into several convenient groups, each of which is defined in the specifications of the work and each of which usually calls for a different contract price. Quite commonly the groups are earth, hardpan, loose rock, and rock, and while these are satisfactory for certain kinds of excavation work they do not cover a wide enough range. The following classification is suggested and will be used in the discussion of methods.

Class Ia.—Sand, fine gravel, soil, and silt containing little or no water. *Class Ib.*—Quicksand, mud, and clay containing more or less water.

Class II.—Sand and soil containing boulders and large stones, coarse gravel, and loose rock.

Class III.—Hardpan, compact hardpan, shales, more or less fissured masses of soft rock, coal, compact and partly cemented gravels, and rock masses greatly weakened by sheeting, joint, and movement planes.

Class IV.—Medium-hard rocks such as sandstone, limestone, slate, volcanic tuffs, volcanic glasses, breccias, cemented gravels, and rock masses weakened by sheeting, joint, and movement planes; greatly altered rock masses, soft ores.

Class V.—Compact hard rocks, silicified limestones, igneous and metamorphic rocks, gneisses, moderately altered rock masses, quartzite, and most ores.

TABLE 83.—WEIGHT OF VARIOUS ROCKS, ORES, AND MINERALS, IN POUNDS PER CUBIC FOOT

Rocks or mineral material	Wt., lb. per cu. ft.	Rocks or mineral material	Wt., lb per cu. ft.
Granite	163.3–169	Iron ore (magnetite)	318
Limestone	148.0–177	Iron ore (limonitic)	245
Basalt	168.0–184	Iron ore (carbonate)	239
Sandstone	116.0–154	Iron ore (Mesabi) loose	148
Clean sand	90	Quartz, solid	167
Clay, dry	100	Quartz, loose, broken	100
Clay, damp plastic	100	Quartz broken fine and well packed	112
Gravel, clean	100	Quartz as quarried	94
Gravel, sand, clay	100	Slate	162–178
Soil	100	Shale	162
Mud, dry	80–110	Shale, broken	92
Mud, wet	110–130	Serpentine	162
Soft rotten rock	110	Soapstone	170
Hard rotten rock	100	Talc	168
Feldspar	162	Trap	187
Flint	162	Anthracite coal, solid	87.5–112.5
Gneiss	168	Bituminous coal, solid	75–94
Greenstone	187	Anthracite, broken	52
Greenstone, broken	107	Bituminous, broken	47–55
Hornblende	203		

Class VI.—Very tough hard rocks, unaltered igneous rocks like granite, diorite, diabase, tough metamorphics such as flint and jasper, dense hematite and magnetite ores, and highly silicious ores not weakened by fractures.

Material of classes I and II requires no loosening as a preliminary to excavation, while material of class III must be loosened by plows or other mechanical devices or by blasting with explosives. Material of class IV requires preliminary blasting with low-grade dynamites, class V requires medium dynamite, and class VI, 40 to 60 per cent dynamite.

Physical Properties of Rocks.—The physical properties of rocks of importance to the engineer are density and porosity, the void spaces in broken or loose material, the extent to which excavated material increases in volume on being excavated, and the degree of consolidation that takes place in banks and piles while and after they are placed in position.

TABLE 84.[a]—PROPERTIES OF FROZEN MATERIAL

		Black and sandy	Gravel and sand	Bedrock
Av. sp. gr. of material frozen		1.401	2.189	2.59
Av. sp. gr. of thawed and dry		2.411	2.69	2.655
Av. sp. heat of frozen material		0.196	0.172	0.183
Av. per cent of ice in frozen material	by volume	68.2	29.1	9.6
	by weight	44.7	16.0	4.26
Av. per cent of solids in frozen material	by volume	31.8	70.9	90.4
	by weight	55.3	84.0	95.74
Av. percentage of voids in frozen material		0.0	1.28	0.0
Av. percentage of voids by volume in thawed material		6.1	3.97	1.65
Av. percentage of ice per cu. ft. of material		39.11	22.00	6.96
Av. lb. of solids per cu. ft. of frozen material		48.39	115.50	159.95

[a] From *Eng. Min. Jour.*, vol. 95, page 706.

Table 85 gives the weights of materials of different specific gravities and for different percentages of void space.

TABLE 85.—WEIGHTS PER CUBIC FOOT

Sp. gr.	2.6	2.7	2.8	2.9	3	Wt. of water in lb. to fill voids
Wt. per cu. ft. solid	162.5	168.8	175.0	181.3	187.5	
30 per cent voids	113.8	118.2	122.5	126.9	131.2	18.75
35 per cent voids	105.6	109.7	113.8	117.8	121.9	21.88
40 per cent voids	97.5	101.3	105.0	108.8	112.5	25.0
45 per cent voids	89.4	92.8	96.3	99.7	103.1	28.3
50 per cent voids	81.3	84.4	87.5	90.7	93.8	31.3

Table 86 gives the weight per cubic foot for different mixtures of sand and water. For an increase or decrease of 0.1 in specific gravity of material, multiply weight of sand by 0.37 and add or subtract this weight from the weight of sand and the aggregate weight given in table.

TABLE 86.—MIXTURES OF WATER AND SAND OR GRAVEL

Per cent of material by volume	Sp. gr. of solid 2.7						
	3	5	6	7	8	9	10
Wt. of sand per cu. ft., lb..	5.06	8.45	10.13	11.83	13.5	15.2	16.88
Wt. of water per cu. ft., lb.	60 62	59.37	58.75	58.12	57.5	56.87	56.25
Aggregate wt. per cu. ft. of mixed sand and water, lb..................	65.68	67.82	68.88	69.95	71.0	72.07	73.13

The determination of density can be made by excavating the material from a known volume and weighing it in its natural state and then drying it and reweighing. Thus two quantities can be figured, the weight per cubic foot as excavated and the dry weight per cubic foot. In porous or spongy ores the volume occupied by selected pieces can be determined by the sand-displacement method and from the weight of the piece the density can be figured. In the case of compact ores the determination of the average specific gravity can be made from a number of pieces by weighing each in water and in air in the customary manner.

Practically all earths, ores, and rocks increase in volume when broken and removed from place. Shaking or ramming reduces the volume and increases the density. With soft materials the original volume is approximated, with spongy material like some iron ores the volume can be made less than the original, while hard broken rock admits of a minimum reduction in volume. Two methods for the measurement of excavation are in use, "place measurement," which is preferable, and "loose or car measurement." The ratio between place measurement and car measurement for different materials is given by the Bucyrus Steam Shovel Handbook in Table 87.

TABLE 87.—RELATION BETWEEN PLACE AND CAR MEASUREMENTS

	Minimum	Maximum	Average
Spongy iron ore........................	0.7	1.07	0.94
Sand..................................	0.56	0.56	0.56
Clay..................................	0.47	0.77	0.61
Earth.................................	0.43	0.77	0.53
Rock..................................	0.12	0.79	0.43

The amount of shrinkage or consolidation after filling depends upon the nature of the material, the presence of water, the action of rain, and the amount of rolling or compacting when placed.

METHODS OF EXCAVATION

The nature of the material, the topography and general conditions, the quantities to be handled, and available capital, engineering, and man power determine methods of excavation and the selection of mechanical appliances. The problem of the open-pit engineer is economical and efficient handling of ore and of waste rock, which may be overburden or may be in part included within the ore mass. Small-scale and pioneer operations may make use of manual methods, pick, shovel, wheelbarrow, cart, mine car and track. Systematic mining calls for the use of modern excavating and transportation machinery and methods. Excavation involves loosening and breaking by machinery or by drilling and blasting, loading, transportation, and dumping.

Material handled in open-pit operations ranges from alluvials, hardpans, clays, and shales up to hard rocks. Usually some preparation is required for alluvials, such as plowing, breaking by means of a rooter operated by tread tractor, or brush-clearing by a bulldozer. Drilling and blasting are a necessary preliminary for the harder rocks. Excavation sites are naturally drained or can be drained by ditches, pumps, or other means; some excavations are best handled by dredging.

Excavation appliances fall into two groups, (1) excavators and loaders and (2) excavators, loaders, and transporters. Certain appliances can be called transporters and others unloaders. Special methods involve dredging, hydraulic mining, trenching, stock-piling, and reclamation, and appliances have been more or less standardized for these.

Excavators and Loaders.—These tools or machines are the power shovel, dragline excavator, continuous-bucket excavator, and locomotive crane and bucket. The power shovel excavates from the level upon which the machine stands to a bank height determined by the highest position of the bucket; it loads into cars, trucks, belt-conveyor hoppers, or other units. The dragline excavator digs below its level but can also carry a bank of moderate height; it can dig under water. The locomotive crane and bucket can excavate below the level and under water and can also excavate from stock piles of low height. Of the three the power shovel can handle large, coarse rock and can excavate tighter and more compact materials than can the others; the dragline excavator is more effective over a wider range of loose rock and alluvials than the locomotive crane and bucket, which is used in a more restricted capacity than the others.

Power Shovel.—The modern power shovel is full-revolving, mounted on tread tractors and operated by electric motors, diesel engine, gasoline

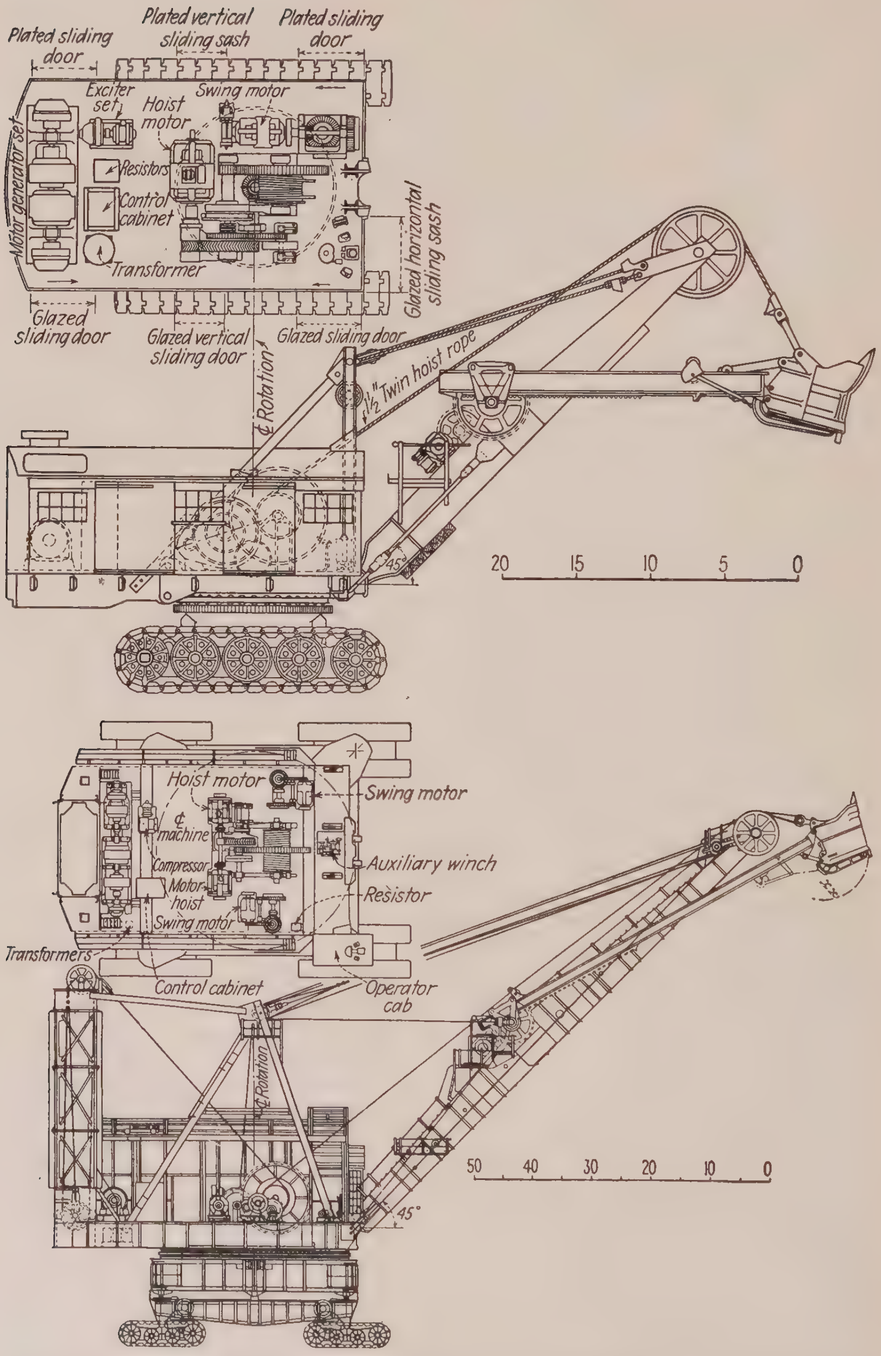


FIG. 189.—Upper figure is a 5-yd. electric shovel; lower is a stripper or long-boom shovel. (*Bucyrus-Erie Co.*)

engine, or less frequently by steam power. It is a mobile unit moving under its own power and handled by one operator. Its principal motions are the swing about its axis, the crowding or thrusting action of the dipper as it takes its load, the lifting action as the dipper cuts into the bank, the final swing to dumping position, and the opening of the dipper door to discharge the contents of the dipper. The dipper is provided with a cutting edge and with teeth for tearing and cutting into the bank and for handling coarse loose rock. Figure 189 illustrates a medium-sized shovel and a large stripper.

Small power shovels range from 3/8 cu. yd. dipper capacity up to 3/4, 1, 1½, and 2 cu. yd.; large, heavy shovels from 2 to 5 cu. yd.; and very large from 8 cu. yd. up to the maximum size of 35 cu. yd. dipper capacity. The very large shovels are used in coal stripping. The small shovels are used in small clay, industrial mineral deposits, gravel, road building, and auxiliary service about large open pits. The large shovels are used in iron-ore and copper-ore open pits.

The 3/4-yd. shovel has a travel speed of 0.9 m.p.h., a swing speed of 4 to 5 r.p.m., a bail speed of 90 ft. per min., and an 84-hp. engine. Its maximum cutting height is 25.5 ft., maximum cutting radius 25 ft., and maximum dumping height 18 ft. It weighs 21 tons. The 5-yd. shovel has a boom length of 35 ft., dipper handle 22 ft., maximum dumping height 19.75 ft., maximum dumping radius 40 ft., maximum cutting radius 46 ft., digging depth below grade 9.5 ft. Power is provided by Ward-Leonard electrical equipment with a 275-hp. induction motor; the propelling speed is 0.9 m.p.h. and the weight is 172 tons. A 32-yd. stripping shovel has a boom length of 102.5 ft., dipper handle 69 ft., boom angle 47.5 deg., maximum dumping height 76 ft., maximum dumping radius 110 ft., cutting height 89.75 ft., maximum cutting radius 119.75 ft., and cleanup radius 63 ft.; Ward-Leonard electrical equipment with a 1,100-hp. induction motor; working weight, 1,550 tons. All the foregoing shovels are equipped with tread tractors.

Shovel capacity depends upon material handled and bank, loading, and transporting conditions. The present tendency in design is to increase power relative to dipper size, thus giving greater speed in swinging, crowding, and hoisting. A 5-yd. shovel may be equipped with a 4-, 4.5-, or 5-cu. yd. dipper; the harder the digging, the smaller the bucket. In loading iron ore upon the Mesabi, electric shovels give average and maximum loading rates per 8-hr. shift as follows: 1¾-yd., 1,600–3,000 long tons; 3-yd., 2,500–5,000; 4-yd., 4,000–5,000 long tons, with power consumption ranging from 0.2 to 0.33 kw.-hr. per long ton. In loading loose broken rock, only about 75 per cent or less of dipper volume may be used, whereas with much fine material up to 90 or more per cent may be used. A shovel may complete three cycles a minute in easy digging, and not complete one in hard digging. The shovel operator's skill is another factor. A factor of 65 per cent theoretical capacity is sometimes taken in estimating working capacity. At Morenci a 4.5-yd. shovel loads some 5,000 tons per 8-hr. shift; at Copper Flat 4- or 4.5-yd. shovels load 5,800 tons per shift. In small operations only a fractional part of shovel capacity is utilized, but in large, well-planned operations shovel capacity may considerably exceed 65 per cent of theoretical capacity.

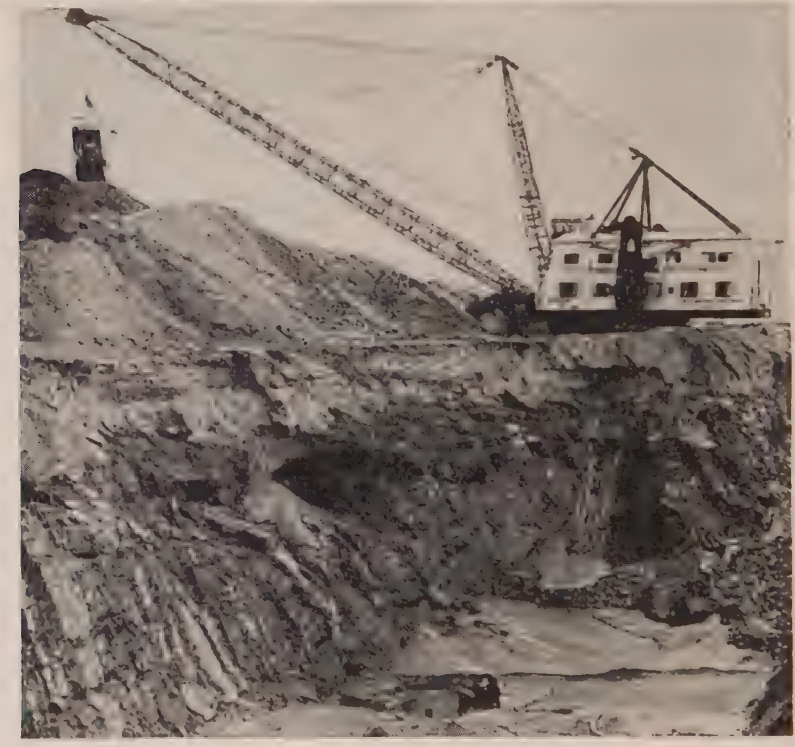


Fig. 190.—Dragline—showing pit and dump. (*Page Eng. Co.*)

Dragline Excavators.—The dragline excavator was developed by John W. Page in 1903 and the Page bucket was patented in 1904. The full revolving steel frame-and-boom dragline was brought out in 1906 and the walking dragline in 1912; in the latter year internal-combustion engines were used for motive power. Since then the dragline (Fig. 190) has been improved, and electric, diesel, and gasoline engine units are used as conditions require. Bucket capacities up to 25 cu. yd. (weight 35 tons—plus load 65 tons) and booms up to 250 ft. in length represent the extreme in size. The boom is mounted on a heavy steel frame that can be rotated 360 deg. The boom angle can be adjusted to digging conditions. The bucket is operated by two lines, one for hoisting and the other for pulling the bucket into its cut and loading it; slacking off on this line dumps the bucket. The revolving frame is mounted upon a large steel drum that gives stability. The walking arrangement consists of steel pads or shoes on the sides of the revolving frame. These pads are connected to vertical frames operated by large-throw cranks carried by a shaft rotated by a large-diameter gear wheel. Steps 6 ft. in length are made at the rate of three per minute, giving a speed of about 1,000 ft. per hour.

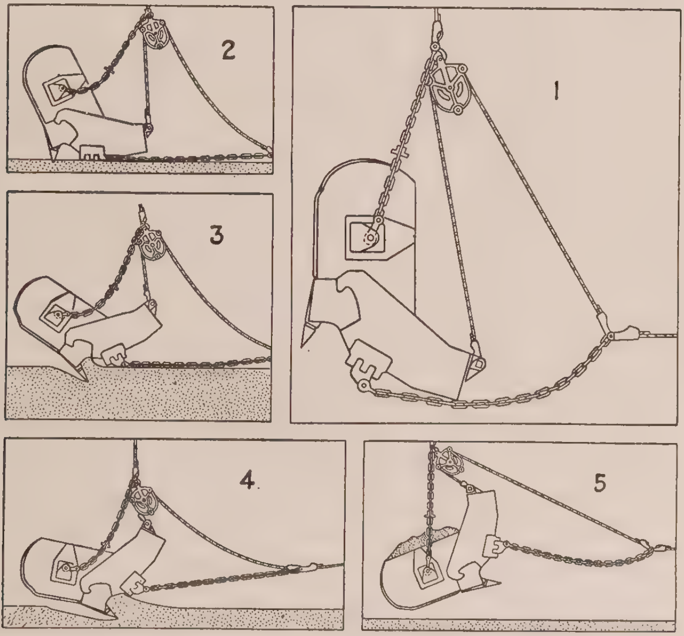


FIG. 191.—Digging action of dragline bucket 2, 3, and 4; lifting of bucket 5 and discharge position 1. (*Page Eng. Co.*)

Buckets vary in weight to suit digging conditions; heavier buckets with the weight concentrated close to the cutting edges are required for hard as compared to easy digging. For example, the 3½-cu. yd. bucket is made in three weights, 6,000 lb., 7,100 lb., and 8,000 lb., and even heavier buckets are made for tight digging. The bucket action is shown in Fig. 191. Manganese steel is used for teeth, cutting edges, and bucket bottoms and sides.

The foregoing description applies to the Page dragline. As the mechanical design of the power shovel is similar, manufacturers of power shovels also manufacture draglines. To convert a power shovel into a dragline requires the attachment of the dragline boom in place of the shovel boom, a change in gears and hoisting drums, and a fairlead for the digging cable. The tread-tractor mounting is equally applicable to the dragline. Designs of walkers other than the Page are also in use. The weight of a 4½-cu. yd., 60-ft. boom dragline is 140 tons; the same machine with a 100-ft. boom will handle a 1.75-cu. yd. bucket. A Marion walker dragline, 5-cu. yd. bucket and 100-ft. boom, has a working weight of 184 tons; a Bucyrus-Monighan walker, 7-cu. yd. bucket and 200 ft. boom equipped with electric motor, has a working weight of 478.5 tons; with diesel-engine equipment the weight is 510 tons.

The dragline is a versatile machine. It will dig noncoherent and well-fragmented blasted material but not coarsely broken or compact materials. Its digging range is 100 ft. below the fairlead and 50 ft. above it. It is a high stacker, depending upon the length and inclination of the boom. It will excavate below water. Short-boom draglines are used for loaders for trucks and cars. Long-boom draglines are used in coal stripping, overburden excavation, in soils, sands, glacial drift, and gravels, for sand and gravel pits, for ditches, channels, and levee building. Phosphate, placer, iron ore, and nonmetallic minerals are within its scope where the materials are of a suitable nature.

Dragline capacity is dependent upon the machine size and type, the operator, and digging conditions. The smaller draglines operate on two cycles per minute; in submerged digging, as in many placer mines, buckets are only 75 per cent filled. Submerged digging is about 25 per cent less than dry digging because of bucket spill. Larger draglines and long-boom rigs operate on a longer cycle; one of 10 or 12 cu. yd. will have a cycle of 1¼ min. for a full swing all the way around; one of 8 cu. yd. on short-swing work may average a 40-sec. cycle; with a bucket that digs poorly the cycle may average 2 min. Moving speeds for walkers are 800 to 1,300 ft. per hr. and for crawlers about 1 m.p.h. for the ¾- to 1-cu. yd., 60-ft. boom machines. Shallow digging and low banks require frequent moves, which cut down digging time; short swings against full swings give greater capacity. In coal stripping, a 9-cu. yd., 150-ft.-boom dragline gave 225,000 cu. yd. per month from a 20- to 35-ft. bank with a 45-ft. cut; a 12-yd., 185-ft. boom gave 450,000 cu. yd. per month on 50-ft. over-

TABLE 88.—TIME CYCLE IN SECONDS—90-DEG. SWING
(Bucyrus-Erie)

	Electric shovels				Draglines			Strippers			
Bucket capacity, cu. yd.	⅜	¾	2½	5	5	8	12–15	17	19	21	35
Boom length	..	..	..	..	100	150	175–200	90	95	95	
Dipper stick	..	..	..	..	...	...	....	67½	61½	56½	
Easy digging	15	18	20	20	43	55	55	42	45	50	45
Medium digging	18	20	22	22	48	60	60	45	48	53	48
Hard digging	24	26	26	28	53	66	68	50	53	54	54

burden. A 5-yd. handles approximately 2,200 cu. yd. per shift; an 8-yd., 3,000; and a 10-yd., 3,600. The foregoing figures are only examples; precise knowledge of digging conditions is necessary for accurately estimating dragline capacities.

Power Equipment.—Large shovels and draglines require special equipment for electrical operation. Variable voltage d-c apparatus or Ward-Leonard control is in general use. There are design variations but the general arrangements are much the same. Power is taken by trailing cables of special construction at voltages of 2,300 to 4,000 a-c. This is delivered to a synchronous or a-c motor, which in the case of the shovel is direct-connected to three d-c generators, one providing current for the hoist motor, another for the crowd or thrust, and the third for the motor or motors operating the swing. A fourth generator, motor-driven or V-belt drive, supplies excitation current to both generators and motors. Stalling of hoist and thrust motors with resultant large current volume and heating necessitates control apparatus for these normal contingencies.

Both generators and motors are of the heavy mill type with insulation designed to stand relatively high operating temperatures. D-c motors give maximum torque at the stalling point, and as speed increases torque decreases. Resistors are used for regulating field currents in generators and motors. Shunt-wound motors permit of regenerative breaking. Three master switches operate contactors which vary the strength and direction of the generator fields, controlling the amount of current generated and applied to each motor, and which also vary the motor fields, giving flexibility of operation. Thrust, hoist, and swing are controlled by these switches. Another method of hookup is the Amplidyne control, which when applied to thrust, hoist, and swing motors reduces the size and rating of switches, resistors, capacitors, and transformers and results in better functioning of all motors. Overpower and underpower peaks are reduced so that there is a smaller spread between the demand curve and the response curve.

The gasoline- or diesel-engine shovel is equipped with a single engine that operates the crowd, swing, and hoist through the medium of well-designed clutches. A roller chain connects the engine and transmission or gear train to the vertical swing shaft, the vertical propelling shaft, and the drum shaft; sprocket chains connect with the thrust or crowd drum, which operates the wire ropes effecting thrust and retraction of the dipper stick. Manually operated band brakes control all movements.

Locomotive Cranes.—These are mobile units used with tracks or equipped with tread tractors. The design follows closely that of the power shovel. Stacking height and radius of operation are determined by boom length and angle. Buckets of many designs can be used, clamshell, orange-peel, Hulett excavating bucket, Brownhoist grab bucket, and others in various sizes. Such cranes are important auxiliaries in open-pit mining, where they are used in track shifting, stock-pile reclamation, unloading, and construction work. With suitable buckete quipment the unit can excavate below the level upon which it stands. As an excavator its range is limited to soft materials, gravels, muds, and underwater material of like nature. It is used for unloading barges and cars.

Continuous-bucket Excavators and Loaders.—Machines of this classification have been used in great variety for excavating work, but being specialized they lack the versatility of the machines already described. Two general designs have appeared, one for digging and loading out a bank extending above the level of the machine, the other for a bank below its level and at one side. Both take a longitudinal slice along the bank. The excavator is mounted upon trucks carried upon tracks that parallel the bank face. Both types consist of a ladder or structural frame with tumblers carrying a chain of buckets. The buckets are drawn against the bank face, scraping out their load, and at the same time the whole machine travels slowly along the face; the buckets are then dumped at the tumbler into a hopper that discharges into the cars beneath. The machine is designed for a given situation, being limited to noncoherent or only slightly consolidated material. It is used in stripping coal deposits and is more often seen in Europe than in the United States. The continuous-bucket loader mounted on tread tractors is a mobile loader similar to the foregoing in mechanical principle, but it is used only for loading sand and small-sized broken rocks into cars or trucks. Trench diggers are also similar in principle to this connected- or continuous-bucket excavator.

The elevating grader with disk plow, inclined belt conveyor, diesel engine, steel frame, and four wheels is drawn by a tread tractor and is used for excavating and for loading trucks. The largest machine of this type weighs 10 tons. Its 48-in. belt conveyor, which can be raised and lowered to loading height, is effective in stripping thin overburdens of relatively soft materials.

Power Scraper Excavator-loader-transporter.—The pioneer of this type is the horse-drawn scraper, manually operated. The fresno and then the two-wheel fresno next appeared. Long after, the Le Tourneau Carryall, a four-wheel rubber-tired scraper, tread-tractor drawn, revolutionized shallow excavating in soft ground. Table 89 gives sizes of power scrapers. Next the rooter, a heavy multiple-breaking plow of simple design, extended the range of the power scraper to harder and more compact materials. The large-diameter rubber tires with which the Le Tourneau scraper (R. G. Le Tourneau) is equipped and the tread tractor or caterpillar used for hauling it contributed largely to the success of this scraper unit. The Tournapull is a four-wheel bowl scraper equipped with its own engine but requiring a pusher tread tractor when being loaded. An 11-yd. Tournapull is equipped with a 98-hp. engine, and a 15-yd. with a 150-hp. engine. The maximum speed is 14.3 m.p.h. Transportation is speeded up with this unit.

TABLE 89.—TREAD-TRACTOR-DRAWN SCRAPERS

Make	Blade width, ft.	Depth cut, in.	Capacity, cu. yd.		Weight, tons	Length, ft.	Width, ft.
			Struck	Heaped			
La Plante Choate:							
2-wheel C41	3.1	8	2.9	3.8	3.86	13.3	7.8
C61	3.8	9½	4.1	5.5	4.04	13.3	8.8
C71	5.0	9½	6.5	8.2	6.04	15.6	10.6
4-wheel C74	8.0	14	9.5	12.0	8.6	27.0	10.0
C86	9.0	18	15.3	19	12.0	31.6	11.5
C104	9.0	18	25.0	33	19.4	39.6	11.5
Le Tourneau:							
NU	10.0	....	33.3	45	22.5		
RU	10.0	....	23.0	30	18.4	42.0	11.6
FU	10.0	....	17.7	23	14.3		
LU	8.5	12	15.0	19	12.9	35.3	10.1
LP	8.5	8	12.1	15	9.8	32.1	10.1

NOTE.—Depth of cut, which is adjustable, is shown at the maximum.

The scraper skims its load, forcing it into the bowl, which is lifted clear from the ground when full and then transported to the place of disposal, ejection of its contents being accomplished by a movable tail gate; operations of loading, lifting of the bowl, and ejection of its load are cable-controlled by the operator of the tractor. Such scrapers up to 33 cu. yd. heaped capacity are in use. Transportation distance ranges up to round trips of 6,000 ft., although the capacity of the unit is greatly diminished at such distances. It is best used for short hauls, 800 to 2,400 ft. Maximum tread-tractor speeds are 4.9 m.p.h. with drawbar pull of 7,995 lb.; six speeds are available, the first being 1.6 m.p.h. with a drawbar pull of 26,208 lb. (D-8 Caterpillar). The D-7 Caterpillar has five speeds, ranging from 1.14 up to 6 m.p.h.

Following the Le Tourneau scraper many other similar units have appeared, among them the tractor-operated two-wheel scrapers, controlled by rope or hydraulic cylinder, in both front- and rear-dump types. These scraper units have found application in stripping operations. They are efficient diggers in sand, alluvials, and the softer hardpans and shales where rooters can be used to prepare the material for scraping. The success of the power-drawn wheeled scraper lies in its rapid loading. At 141 ft. per min. the smaller scrapers operating at a cut of 6-in. depth can load in ½ to ¾ min.; the larger sizes in ½ to 1 min. The largest (Carrimor) have a width of cut of 10 ft. and a depth of cut adjustable up to 18 in. With the deeper cuts loading time is correspondingly lowered.

Wire-rope-operated scrapers and buckets have been utilized in mining operations for many years. Usually a three-drum hoist is used, one drum for the pull-in rope and the others for tail-rope operation. By the use of two tail ropes a fan-shaped area can be focused upon the loading ramp. The hoist is placed close to the loading hopper and the scraper ramp leading thereto. Tail ropes are carried by sheaves that can be moved to new positions. Operating distances are short, 500 to 1,000 ft. Loading is into truck or car by overhead hopper. Such scraper arrangements have been used in placer operation, for stripping overburden, and for excavating clay where it is not too compact. Only readily excavated materials can be handled, though preparation by a breaking plow may extend its scope.

The cableway type of power scraper appears in several forms. The simplest is a two-rope system in which the pullback or tail cable is supported by towers and the bucket is some form of the bottomless scraper with cutting edge and teeth like the slusher. A two-drum hoist is used. The head tower gives sufficient height to serve the loading hopper. With a movable tail tower a fan-shaped area is served; when both towers are movable a rectangular area may be covered. In the gravity-return scraper rig, a track cable (slack line) is supported by two towers on either side of the excavation. The head tower is of such a height as to provide for the return by gravity of the bucket attached to its carrier from its discharge to its digging position. A radial coverage is secured by a movable tail tower. The rig digs, transports, elevates, and dumps into a loading hopper or waste pile. A two-drum hoist provides for tensioning the track cable and handling the pull-in rope. A stop button on the track cable establishes the dumping position. The span served ranges from 200 to 1,200 ft.; digging speed is 200 ft. per min. and inhaul speed 600 ft. per min. Power is electric motor, diesel, or gasoline engine. The buckets, ranging up to 3½ cu. yd. or larger, are equipped with digging teeth and are similar to the dragline bucket. Power consumption, according to H. A. Roe, ranges from 0.6 kw.-hr. per ton for a 200-ft. haul up to 1.7 kw.-hr. per ton for an 800-ft. haul.

Bulldozer.—The bulldozer is widely used for many purposes about open pit, quarry, and mine plant. It consists of a tread tractor upon which is mounted a broad blade which can be raised and lowered and which can cut a wide swath in front of the tractor. The bulldozer pushes and crowds the earth pile ahead. It is invaluable in clearing brush and surface rubbish and can also excavate and move lighter alluvials and earth. Small-scale stripping in placer operations, clearing outcrops in prospecting, foundation excavation, road construction and road clearing in open-pit mining, and waste-rock-pile leveling are among its uses. By means of a ramp and hopper it can be used as a loader. Various attachments have been devised, among which is a bucket that converts the tractor into a mobile digger and loader. The Athey stockpile loader, used in car loading, is a tractor-bucket combination.

Transportation.—Short haul transportation is by car, wagon, and light autotruck in small-scale operations. For medium-scale operations limited to several miles, the heavy truck and the tread tractor with trailer find wider and wider applications in mining. The tread tractor is a form of road locomotive operable over rough roads and is a part of the equipment of most open-pit mines, placers, and mine plants. Table 90 gives sizes and weights. In combination with the Athey trailer, equipped with forged-track wheels, it is used for short open-pit hauls. Both bottom-dump and two-way side-dump trailers are supplied in capacities from 11 to 16 cu. yd.; loading height 6 to 7 ft.; gage 74 to 78 in.; turning radius 12.5 to 17 ft.; and weight 8.8 to 14.5 tons. Maximum tractor speed is 5.9 m.h.p.

TABLE 90.—CATERPILLAR TRACTOR SIZES

Details	D8	D7	D6	D4
Weight, tons	17.08	11.95	8.32	5.12
Drawbar hp	113	80	55	35
No. speeds	6	5	5	5
Minimum speed, m.p.h	1.6	1.4	1.4	1.7
Drawbar pull, at minimum, lb	26,208	21,351	14,300	7,852
Maximum speed, m.p.h	4.9	6.0	5.8	5.4
Drawbar pull, at maximum, lb	7,995	4,550	2,650	2,230
Tread gage, in	78	74	74–60	60–44

For medium-length hauls and open-pit capacities from 1,000 to 5,000 tons output per day, motor trucks are used where road surfaces can be maintained. These are available in great variety, ranging from 5 to 40 tons in capacity. Figure 192 shows a truck design used by open-pit mines. Trucks are operable over adverse grades up to 14 per cent and have proved to be so versatile that they have displaced track and train transportation in numerous open-pit operations and in coal strip pits. Trackless pits are now common, especially where open pits are deep and restricted in area. More cheaply constructed pit approaches are possible with truck operation.

End-dumping, side-dumping, and bottom-dumping trucks are available. Truck bodies are made of steel, usually high-tensile, and weight is reduced by careful design. Tilting is by hydraulic cylinder and ram, but where trucks are dumped at a fixed place an auxiliary hoist placed above the dumping position is sometimes used for tilting. End-dumping bodies are made scoop-shaped to avoid end gates. At the Mahnomen mine, trucks have the following specifications: Four wheels, with 13.5- by 24-in. dual tires on rear and 12- by 24-in. tires on front wheels; gross load capacity 62,000 lb.; pay load 36,000 lb.; rear-dump scoop-shaped body of 9-yd. level capacity, raised and lowered by a double-acting hydraulic cylinder; 185-hp. oil-burning engine; eight forward speeds from 3.3 to 31.4 m.p.h.; fluid coupling between engine and single-plate clutch; air brakes and usual auxiliaries. The cab is offset to left to give better vision in backing; the small wheel base is an advantage in pit operation. At Morenci, 22½-yd. trucks, diesel engine, pay load 33 tons, six wheels with dual 13.5 by 24-in. tires on rear, were used in early pit development and gave costs of 6.1 cents per ton-mile over average hauls of 0.6 mile; truck units cost $35 per shift for fuel, operation, repairs, and supplies, and tire wear exceeded 12,000 miles. Poor road surfaces and abrasive material in the pit materially increase tire wear.

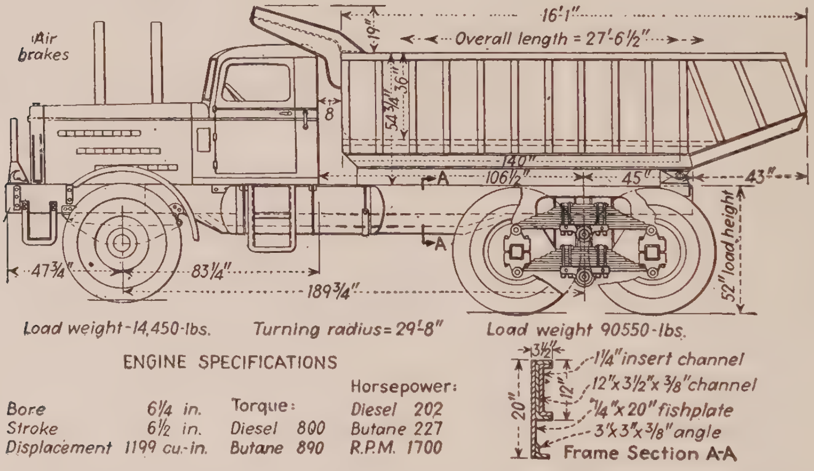


FIG. 192.—Type of truck used in some open-pit mines. (*Eng. Min. Jour.*)

For long hauls and large tonnage, track and train equipment has been a feature of pit operations for many years. Steam locomotives, which are limited to adverse grades of 3 to 4 per cent, are being displaced by electric locomotives and diesel electrics that can operate on adverse grades up to 8 or 10 per cent, although such grades are avoided where possible. Larry cars and trailer cars are used to some extent and have weight advantages over the electric locomotive. The larry car is equipped with electric motors and thus there is a saving in gross loads as compared with equivalent capacity in electric locomotive and cars. An interesting application is the Woodford third-rail automatic control system. Each car is equipped with d-c motors that provide for regenerative braking control on downgrades. The movement of individual cars is remote-controlled. The system has been successfully applied at some iron-ore pits and limestone quarries. Standard-gage track with heavy rails on main approaches and lighter rails on benches is installed in open pits where train haulage is in use. Hopper-bottom and box-type cars for rotary dump are usual equipment. Ore cars are from 60 to 100 tons capacity. They are taken to the pit benches, loaded by power shovel, and then hauled to the train yards. Modern cars are made of high-tensile steel and are of lighter weight than older cars; for example, a 75-ton car weighs 52,500 lb. as against 84,500 lb. for the older type. In stripping operations, 20-, 30-, and 40-yd. cars of the side-dump, compressed-air-operated type are widely used in copper and iron open-pit operations.

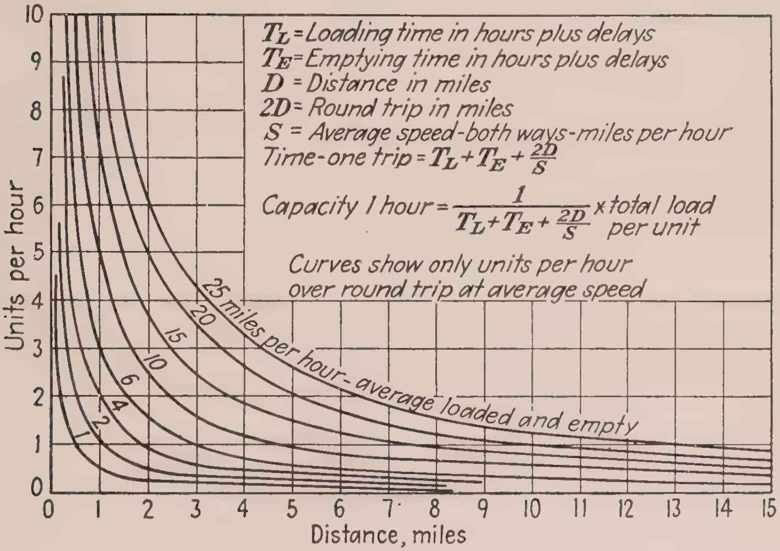


FIG. 193.—Chart showing relation between units per hour, distance, average speed; equation for capacity is also given.

Time to load, to transport, and to dump and the capacity of the vehicle are important factors determining over-all capacity in stripping and ore mining. Distance and the average speed for loaded and return trips determine transportation time. Outward speeds are frequently limited by adverse grades and are usually lower than for return trips when grades may be favorable. The relation between distance hauled and average m.p.h. for loaded and empty trips is shown in Fig. 193. The simple equation given in the figure enables capacity per hour to be calculated when loading and dumping time, distance, and average speeds are known. A unit may be a car, truck, or train, with time factors to correspond. A study of the curves will show why distances over which stripping and ore are hauled are kept to a minimum and why speeds as high as practicable are employed. It will also indicate that where distance is relatively great, the size of units (car, truck, or train) selected should be large where tonnage must be moved. An increase in the size of a unit increases its first cost and calls for heavier locomotive equipment at increased cost. The number of units to serve a shovel depends upon the loading capacity of the shovel and the capacity of units to keep it fully employed; the number of trains to serve a pit depends upon the shift tonnage to be mined and the capacity of the trains. Sufficient excess units are necessary to provide for servicing trucks and for contingencies and delays when trains are used. Train schedules with passing points are systematically made out where a number of trains are employed and the distances are more than nominal.

In "stream-flow" transportation, by belt conveyor or by water in pipe, ditch, or flume, loading and dumping time disappear and transportation time is a function of belt speed or, in the case of water, of velocity and grade. Capacity with belt conveyors is the product of load per foot and velocity, or feet per minute; with water flow, of the proportion of solids per cubic foot and the total flow in cubic feet per minute.

Dumping.—Various methods for the dumping of transportation units are in use. Trucks are usually end-dumping, the body of the truck being raised by hydraulic cylinder and ram. To avoid end doors on trucks the truck body is made scoop-shaped. The side-dumping spoil car has displaced most other types. Box-body ore cars that require rotary dump are being displaced by cars of the side-dumping type similar to the spoil car. Bottom-dump gondolas are standard railroad equipment and are used in the Mesabi open pits. Bottom-dump trucks and trailers are also used, but the end-dump type is more popular.

Special Methods of Excavation.—Subaqueous excavation is a feature of gold dredging. The continuous-bucket dredge used for this purpose will be described in the chapter on Alluvial Mining. Dredges of this type have been used in gravel and sand pits. The hydraulic dredge has been used in gravel and sand pits. A form of dredge using the hydraulic elevator idea has also been used for sand and gravel operations. The bucket dredge has been used in phosphate rock mining. Hydraulic excavation by means of a stream of water under high pressure has been used in coal stripping, nonmetallic mineral stripping, and phosphate rock mining. It is widely used in placer operations. In connection with hydraulic stripping, dredge pumps are used to drain strip-pit sumps and carry excavated material suspended in water through pipes to disposal points.

A classification into six groups of materials is made near the beginning of this chapter. Methods applicable to these are as follows: Class I*a*, all excavator-loaders and excavator-loader-transporters; Class I*b* may involve the same equipment, but where water is excessive and cannot be drained, the dragline excavator, locomotive crane and bucket, wire-rope-operated scrapers and buckets, and the dredges of various types are used; Class II can be handled by power shovels, dragline excavators, and, under favorable conditions, by scrapers and wire-rope scraper rigs; Class III is best handled by power shovels, but after more or less blasting the dragline excavator is adequate, and after preparation by scarifiers and rooters, scrapers of the Le Tourneau type may be successfully used; Class IV must be drilled and blasted and can then be handled by shovel, though the softer materials can sometimes be handled by dragline excavator; Classes V and VI must be thoroughly drilled and blasted before being handled by power shovels. Hydraulic excavation is applicable to Classes I and II and sometimes to Class III when the material is not too compact. However, before selection of digging, loading, and transportation equipment and methods of operation, studies of the materials they are proposed to handle must be carefully made, and often borings are necessary to determine the variations in the physical nature of the material. The area and the depth—maximum, minimum, and average—also are involved in the selection of equipment, as some tools are more efficient in thin, sheetlike deposits and others in thick deposits admitting of high banks.

OPEN-PIT MINING

Application.—Open-pit mining is applicable in copper, iron, coal, industrial minerals, limestone, rock, gravel, and sand and to a lesser extent in gold, silver, lead, zinc, and other deposits. Utah Copper, Copper Flat, New Cornelia, Morenci, Chino, and Chuquicamata are great open-pit copper mines. Out of 230 iron mines, 49,591,309 gross tons were produced by open-pit mines in 1940, as compared to 24,104,590 gross tons by underground mines. In 1938, 465 strip pits produced 30,406,855 tons of coal.

Where general conditions indicate pit methods, a topographical survey is a necessary preliminary, as are test pitting and boring to determine the limits and thickness of the deposit, its value, and the thickness and amount of overburden. Drainage conditions and the physical nature of the materials and enclosing rocks are closely studied. From these the layout of the pit and the mining and transportation equipment necessary are approximated sufficiently to enable cost comparisons of several operational methods to be made. The access point to railhead or road, drainage, pit approaches, disposal of waste and overburden, drilling, blasting, loading, and transportation are considered separately in relation to the general plan.

Topography.—Most open pits are in flat or rolling topography of moderate accentuation; some, however, are in rugged country or on hill or mountain slopes; many have natural drainage but others become sumps. Shallow pits are 50 to 100 ft. in depth, deep pits, 100 to 200 ft., and very deep pits, 200 to 500 ft. The support of the enclosing walls is a problem in the deep pits. The thickness of the deposit and its environment of surrounding hills or ground masses and their rock character determine how great a quantity of marginal material must be removed to secure stable slopes (see Fig. 194). The ratio of total overburden and marginal rock to total tonnage of valuable mineral is a vital element in cost. Too great a proportion of waste may result in prohibitive costs per ton of marketable mineral.

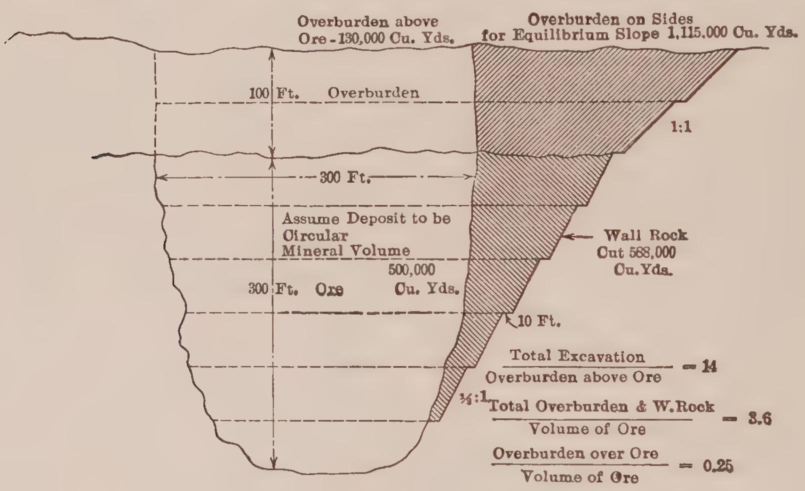

Overburden above Ore - 130,000 Cu. Yds.
Overburden on Sides for Equilibrium Slope 1,115,000 Cu. Yds.
100 Ft. Overburden
1:1
300 Ft.
Assume Deposit to be Circular Mineral Volume
500,000 Cu. Yds.
300 Ft. Ore
Wall Rock Cut 588,000 Cu. Yds.
10 Ft.
Total Excavation / Overburden above Ore = 14
½:1
Total Overburden & W.Rock / Volume of Ore = 3.6
Overburden over Ore / Volume of Ore = 0.25

FIG. 194.—Sectional view of ore deposit.

Scale of Operations.—Small operations may be on the scale of 100 tons of mineral product per day; large operations from 1,000 to 5,000 tons per day, and very large in excess of 10,000 tons per day. Utah Copper produced in excess of 100,000 tons per day and excavated waste rock to 40,000 tons per day; Mesabi iron pits range from 1,500 to 50,000 tons per day during the seasonal operation. Limestone pits range from 500 to 5,000 tons per day; clay pits, from 50 to 200 tons or more. The movement of large daily tonnages presupposes good engineering and well-selected equipment.

Initial Operations.—Open-pit operations begin with the removal of the burden over the area where mining is to start. Stripping may begin 1 or 2 years before mining becomes practicable. Pit and stripping approaches are constructed. Both overburden and mineral mining begin from the top downward in successive horizontal slices (benches) until the bottom has been reached, or until it is no longer practicable to continue open pitting. The walls may be a menace in deep pits. On the other hand, coal strip pits are usually shallow; stripping and mining are carried on simultaneously, the stripping shovel being sufficiently in advance of mining or coal loading so as not to interfere. Stripping is deposited in the previously mined cut as backfill.

Thick overburdens and thick deposits, sometimes characteristic of copper and iron pits, require numerous slices or benches, many of which may be worked simultaneously. Backfilling is seldom possible.

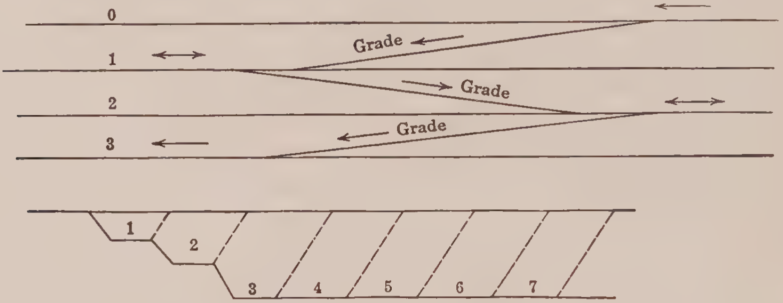
Construction of shops, power plant, supply buildings, and living quarters, as well as railroad or road approaches, is a necessary preliminary. Initial equipment for stripping is first installed. While stripping is proceeding, ore-mining approaches are prepared for the first mining slice, transportation and power facilities and bench tracks are installed, and equipment is assembled.

Pit Approaches.—A thick deposit with a thick overburden may occupy a hill top or ridge, it may be along a hill or mountain slope, or it may be beneath a plain or moderate slope. In the first two cases, benches in overburden and in mineral are reached by switchback systems where tracks are to be used or by road grades where truck transportation is favored. Benches are developed along contours by small power shovels and bulldozers until sufficient width has been obtained for heavier equipment to be installed and tracks to be laid to waste-disposal dumps.

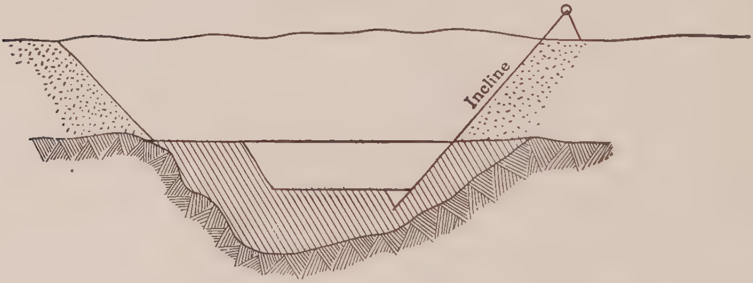
TABLE 91

Vertical depth, ft.	Grade, per cent; length of approach, ft.				
	2	4	6	8	20
10	500	250	166	125	50
20	1,000	500	333	250	100
30	1,500	750	500	375	150
40	2,000	1,000	666	500	200
50	2,500	1,250	833	625	250
60	3,000	1,500	1,000	750	300
Approx. quantity excav., cu. yd., for 60-ft. depth	450,000	225,000	149,850	112,500	45,000

A flat site necessitates an approach with adverse grades. Such approaches may be a railroad grade, an incline of 20 deg. with hoist or belt-conveyor equipment, or a roadway, depending on the grade required. Railroad grades are extended downwards as a box, drop, or pioneer cut, until the first mineral slice or bench has been reached, when they are extended through on a level, assuming that sufficient stripping has been accomplished to make this feasible. Successive slices are reached by extending the grade, often within the pit. Grade approaches (see Table 91) must begin at a considerable distance away from the pit where great depth is necessary; they often involve much excavation. Separate overburden approaches are usually provided so as to separate the two operations, and such approaches are stemmed to the disposal dumps.



(*a*) System of Gaining Depth by Switch Backs



(*b*) Use of Incline and Hoist

Fig. 195.—(*a*) Switchback system; (*b*) incline and hoist.

In copper and iron-ore pits there is an important advantage in being able to take gondolas or large ore cars into the pit for loading; consequently railroad-grade approaches are favored and maximum grades are kept as low as possible. In narrow, difficult pits (Fig. 195) the shaft with skip hoisting or the incline with hoist or belt-conveyor equipment is preferred. With the hoist installation, pit cars are hoisted up the incline and dumped into loading bins; with the conveyor equipment, a grizzly, crusher, and apron feeder are installed below the lowest position of the ore and are served by vertical transfers from the slice being mined, trucks or cars being dumped into the hopper of the transfer. At the top of the incline the belt conveyor discharges upon the shuttle conveyor of the receiving bins, which are also loading bins. In skip hoisting, a short tunnel connects the pit floor with the shaft, and the pit cars are dumped into the skip-loading bins. Other difficult pits are shown in Fig. 196.

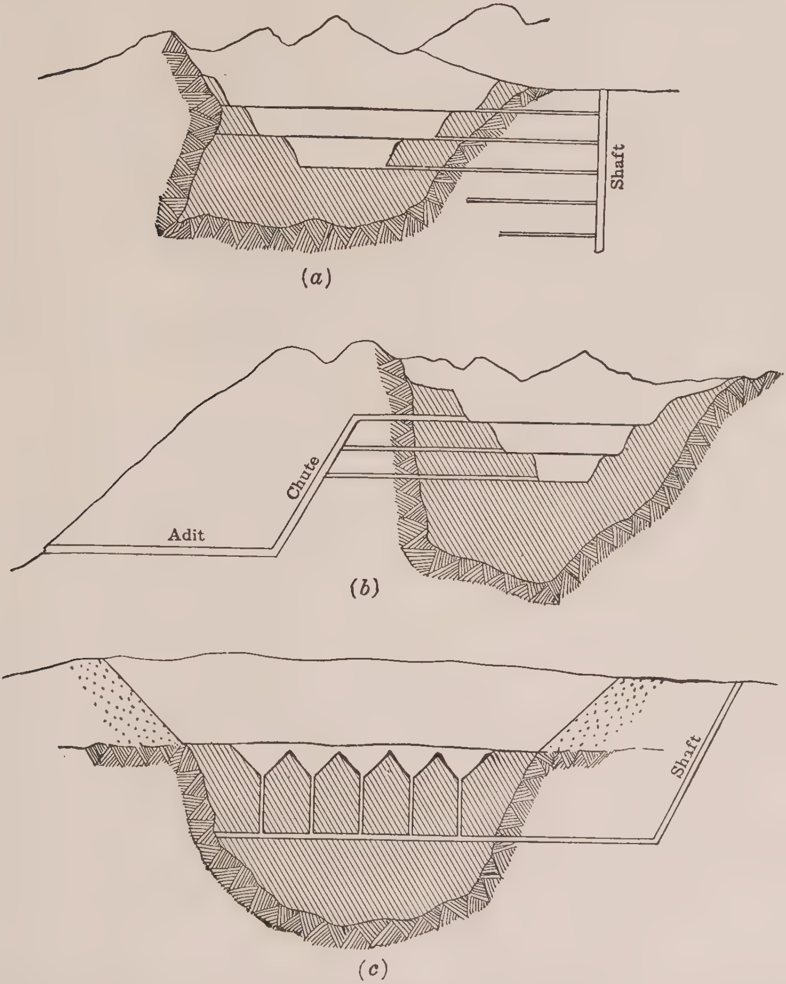


Fig. 196.—(*a*) Access to open pit by shaft and crosscuts; (*b*) access to open pit by adit; (*c*) "milling system" of open-pit mining.

The convenience of truck transportation and the ability of the truck to operate upon grades as high as 12 per cent or more have caused them to be adopted by many operators of pit mines of moderate output. Approaches are shorter than with rail equipment, much less excavation is required, and sharper curves can be used.

Pit Layout.—Initial cuts in an ore slice are made along the longer axis of the ore mass. Because of the position of the ore mass, they may have to be placed on one side. As far as may be practicable, cuts are made straight so that loading tracks may be straight and connecting curves from approach tracks not too sharp. Several benches may be worked when operations on upper benches are kept sufficiently in advance of lower bench operations to cause no interference. Ramps connecting benches provide truck roads for service vehicles. Switchbacks are sometimes placed in the pit to reach lower benches, preferably in unproductive parts of the pit sides. Trackage and haulage distances are kept to a minimum.

The deeper pits require lower slopes and berms, both upon overburden benches and ore benches. Contiguous earth masses are a factor in great open pits. The removal of immense tonnages from such pits unbalances earth masses and causes readjustment that may result in serious slumps in side walls. Such a slump took place in the upper benches of the Utah Copper pit some years ago. When side-wall slumping begins there is nothing much that can be done except to remove equipment and men and restrict mining to safer portions of the pit until the readjustment is completed. It is interesting to note that breaking becomes easier as mining of ore masses proceeds, probably because of the expansion resulting from removal of large amounts of overburden and ore.

Bench Heights.—At one time benches were limited by drilling depth but the modern churn drill has removed such restrictions. A primary limitation is the shovel-digging height; bench heights in excess of this sometimes result in overhangs, which endanger shovel operation by caving. Overhangs seldom develop in most hard-rock pits but may occur where softer materials are excavated. The number of levels in a mine and the number of benches in an open pit have a similar economic relationship. Each bench requires tracks or roads, which must be shifted as successive cuts are taken; and each bench must be drilled and blasted. The lower the bench height, the greater the number of drill setups, with a consequent increase in the handling of tools and explosives and more minor operations, as well as a greater number of moves in loading. In coal strip pits there is only the bench established by the thickness of the overburden above the coal seam and the thickness of the coal seam. The thickness of the deposit establishes bench height for many industrial mineral deposits. In iron mining, contiguous beds of iron ore of different quality are mined separately, and their thickness establishes bench height within limits of 20 to 40 ft., with 30 ft. favored. In copper mining, bench heights are from 30 to 70 ft., with 40 to 50 ft. favored. The Morenci pit was planned with 50-ft. benches in both stripping and ore. In thick limestone deposits, benches up to 200 ft. may be mined. Bench width varies from 50 to 250 ft.; bank slopes range from 45 to 65 deg., the steeper slope being in hard rocks; berms may be of a width one-half of the bank height above.

Box or Drop Cuts.—These are extended downgrade. They may be made by a single shovel cut by a large (8-yd.) power shovel with great swath and sufficient dumping height to load cars upon a track paralleling the cut and upon the level upon which the cut starts. Usually, however, they are made in several cuts, the first being made on a 2 per cent grade with loading out at the side into cars standing upon tracks on the bank above. When the loading height limit is reached the cut is continued upon a level. On completion of the first cut, the tracks are shifted to the drop cut and the shovel takes out another cut of 7 or 12 ft., similar to the first but offset from it. The third drop cut is still further offset. The fourth cut is made close to the line of the initial cut and reaches bench height, or 48 ft. with the bench wall developed. The approach to the new bench is on a grade 2 per cent or steeper as circumstances require.

Switchbacks, Roads, and Ramps.—Switchbacks are used to gain altitude or depth without sharp turns. The switchback is a series of grades requiring alternate advancing and backing of trains. Each grade is made as long as possible and terminates in a tail track of sufficient length to accommodate a train length; the successive elements of the switchback are upgraded in opposite directions, so that all the elements of the switchback cut across contours but are contiguous to each other. A switchback system with components of 1,000 ft. in length will gain 40 ft. on a 4 per cent grade on each component. To overcome an altitude of 400 ft. would require 10 components and 10,000 ft. plus trackage sufficient for nine tail tracks; its horizontal spread normal to the tracks would be about 400 ft. Tail tracks on grades adverse to loads are level or on reduced grade, but with grades favoring loads the tail tracks are upgraded to overcome train momentum and to give quick starts in the reverse direction. The site of the switchback is usually selected to command overburden benches, away from the ore body, but to gain lower benches switchbacks are sometimes placed in the pit on ore banks. Tail tracks connect with the bench tracks. A switchback system is laid out with respect to the benches to be served, needless to say.

A similar system of grades is employed to gain or lose altitude by roads; but each component may be connected by curves, which are compensated and banked to give utmost truck control at such critical points. Curves are 40 to 50 ft. in radius or even less.

Ramps are steep grades connecting benches in a pit, usually for auxiliary truck service. Where they are used for truck service the grades are often 12 to 14 per cent, or greater than those normal to the pit. Spiraling down in a pit (Fig. 197) involves a continous grade on ore banks left at the periphery of successive slices, thus necessarily restricting the minable area in the pit bottom as greater depth is attained. At the finish of normal pit operation, residual ore banks for grade purposes often contain considerable tonnage. They are mined by various methods, some requiring underground development.

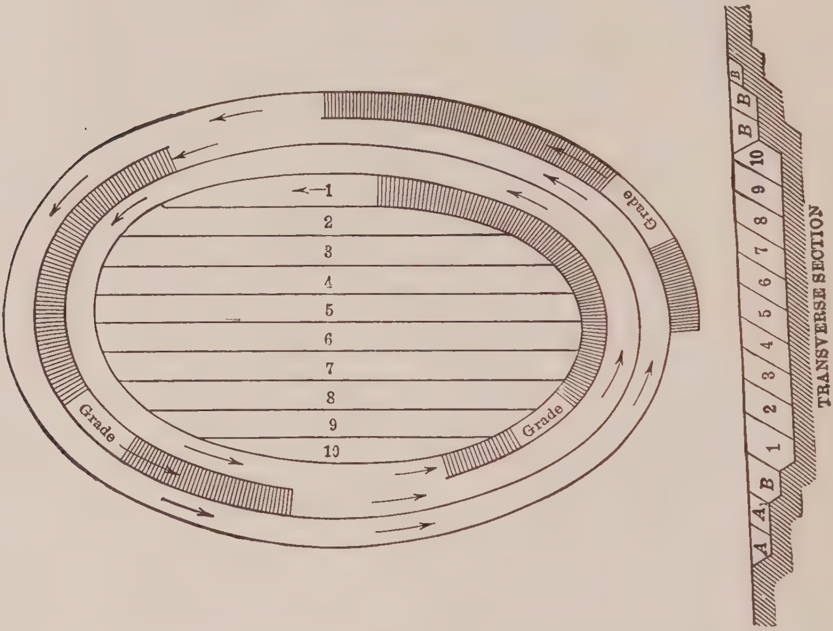


Fig. 197.—"Spiraling" to an open-pit floor.

Main-line tracks in approaches are constructed with 60- to 100-lb. steel, ballasted. In large pits the main lines are double-tracked. Adverse grades are kept as low as possible, to 2 to 3 per cent, and curvatures are from 15 to 30 deg. Lighter steel, 60 to 75 lb., is used on the benches. The foregoing data presuppose heavy train equipment. In very small pits where small cars and light equipment are used, 16- to 20-lb. tracks in a radial track arrangement stemming out from the pit entrance were employed; the bench was developed as a circular wall with individual track heads normal to the wall and about 25 ft. or more apart. However, the increasing use of trucks and the small loading shovel has made tracks obsolete in such pits.

Pit Operation.—Mineral deposits and overburden that can be excavated without blasting are sliced in parallel cuts so as to maintain straight loading tracks, which are placed at the side of the cut in progress.

Bench heights are limited to shovel reach. On completion of a cut, loading tracks are shifted to a position for the next cut. When sufficient clearance has been secured on the first slice, preparations are made to drop cut into the next slice below. It is not always possible to maintain straight loading tracks, and curving benches are quite common in large pits.

Blasting Practice.—There are many variations in bank blasting, but two major practices may be outlined. One involves occasional blasting of many churn-drill holes in a major blast with resultant tonnages to ensure a long period of loading operations; the other requires the periodic blasting of sufficient drill holes in a group to ensure steady loading operations over short periods. Major blasts are often used in initial development of pit benches but are usually avoided after benches have been well established. Chamber blasting is being rapidly superseded by churn-drill holes 7 to 10 in. in diameter with column charging. Detonation fuse (Primacord) is displacing electric blasting caps. Fragmentation is obtained by higher power explosives and by closer spacing of drill holes. Overhangs are avoided by distributed charges, with some of the explosive placed nearer the top of the hole. The use of black powder has greatly diminished. Lox seems to be restricted to some strip-pit coal mining. Secondary blasting has been reduced in amount by the use of large-size crushers and better fragmentation. More or less secondary blasting is necessary in every open pit where blasting is practiced.

On the Mesabi, explosive ratios in open-pit mines range from 4 up to 15 long tons per lb., with powder grades from 25 to 60 per cent in chambering and 25 to 40 per cent for main blasts, the higher grade being used in rock and the lower in ore. In open-pit copper mines ratios are from 4 to 8.5 tons per lb.; powder grades are 40 to 60 per cent for chambering and 30 to 65 per cent for main blasts. Drilling and blasting costs from 2 to 4 cents per long ton in iron mines; from 3 to 14 cents per ton in copper pits.

In blasting, which ranges all the way from light blasting to shake up the ground in advance of loading to heavy blasting with special attention to good fragmentation, each cut is drilled and blasted sufficiently in advance of loading to maintain a bank of broken mineral or overburden, as the case may be. Sometimes an entire cut is blasted in advance of loading, the loading tracks having been removed. The loading tracks are then replaced and the broken material is loaded out while another cut is being drilled. Loading tracks are handled by a track shifter, or by a locomotive crane that handles 33-ft. sections. The removal of loading tracks before blasting is common practice where much fly rock results from blasting.

Churn drilling is used for primary blasts. In toe-holing benches the air-hammer drill on wagon mounting or tripod is favored. For secondary blasting, hand-held drills with detachable bits are standard. Portable air compressors make less necessary the maintenance of compressed-air pipe lines in the pit, although water pipes or water service trucks are necessary for drilling water. Auxiliary trucks serve churn and other drills and bring blasting supplies. Blasting, systematized and under one foreman, is at the end of shifts, and benches are made safe by trimming before loading begins.

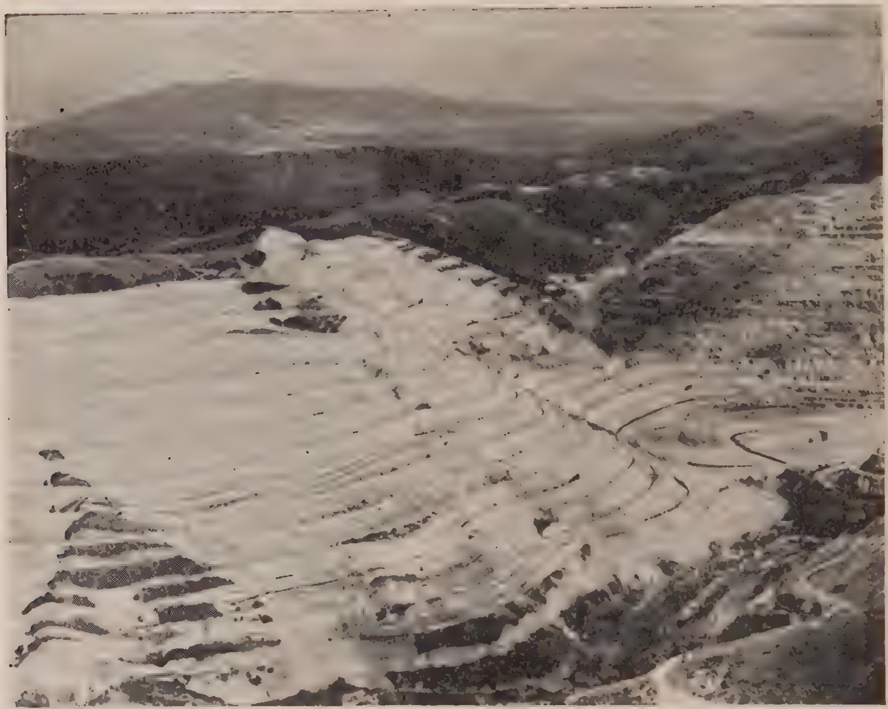
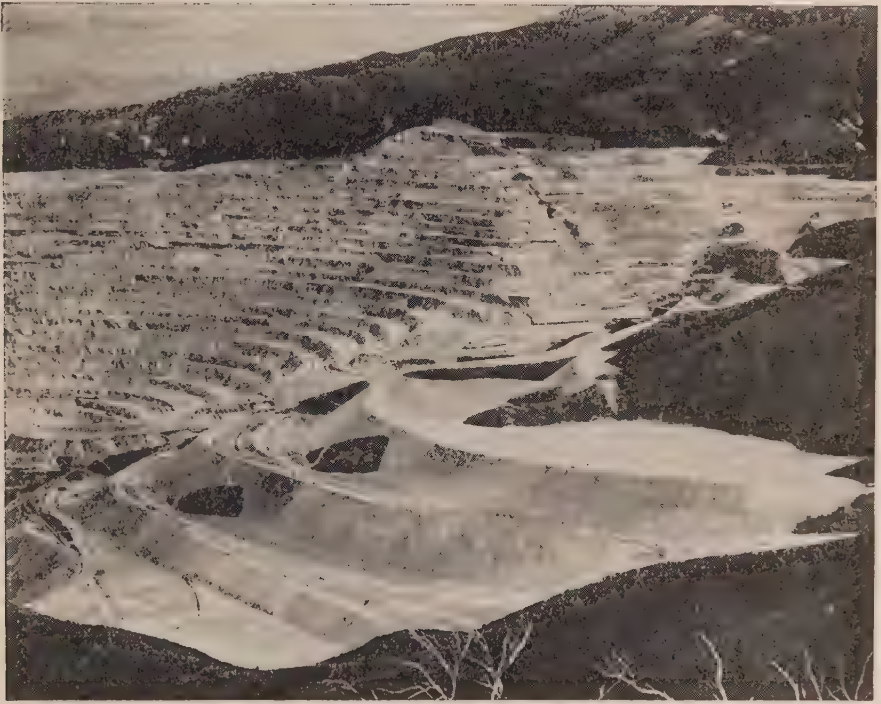


FIG. 198.—Utah Copper mine open pit. Total capping removed as waste rock was 7,903,000,000 lb. to the end of 1944. The

The number of loading units and the number of cuts in progress upon various benches are planned to maintain the desired tonnage output. At the Utah Copper pit (Fig. 198) there are 26 producing benches, each with its shovel and accessory equipment, the aggregate daily tonnage being 100,000 tons or more; at the New Cornelia pit 20,000 tons per day is produced from three or four benches. At the Utah Copper, trains consist of 12 100-ton cars and 14 80-ton cars drawn by 75-ton electric locomotives on 90-lb. rails, mine and dump tracks aggregating 84 miles; at New Cornelia trains consist of six 20-yd. cars or five 30-yd. cars drawn by 67-ton steam locomotives on maximum adverse grades of 2 per cent on 70- and 90-lb. tracks, aggregating 15 miles in the pit and 7 miles outside.

In trackless pits where trucks are operated a similar program is followed but track shifting is replaced by road building, and loading places are cleaned up by bulldozers after each truck has been loaded and before the empty is placed. Road surfaces are kept in good condition and road work displaces track work. In the transporting of sharp rock or highly abrasive materials much attention is required to minimize tire wear.



527,000,000 tons; total ore mined was 453,000,000 tons, and total copper produced was picture was taken in 1944. (*Utah Copper Co.*)

Cars loaded are reported by trainmen. Periodic surveys are made of bench positions and are plotted on pit maps. Consumption of drill steel and weights and kinds of explosive are reported by shifts. Supplies to various units are reported periodically. The pit superintendent reports at intervals upon labor, labor distribution, supplies and repair parts, power consumption, and other details that give an economic picture of operations and enable cost control to be maintained.

The mobility of large pit operations and the use of electrically operated shovels and churn drills as well as electric locomotives necessitates power distribution that can be moved as need arises. Power shovels are served by trailing cable (Fig. 199) as are churn drills. Cable outlets, power lines, and transformers are arranged to conform to bench operations with minimum power losses. Locomotives are also equipped with trailing cables so as to be independent of trolley operation when close to loading shovels. Storage-battery locomotives obviate the use of trolley poles or trailing cable on loading tracks. The diesel-electric locomotive obviates the use of all trolley poles. However, temporary trolley structures serve satisfactorily on loading tracks, though at increased maintenance cost. At the Utah Copper pit, steel poles carry power lines and also brackets for the support of trolley lines; poles are carried upon steel skids so as to facilitate shifting to new positions. The combination storage-trolley type of locomotive is used at Morenci. Electric service is an important part of pit operations.

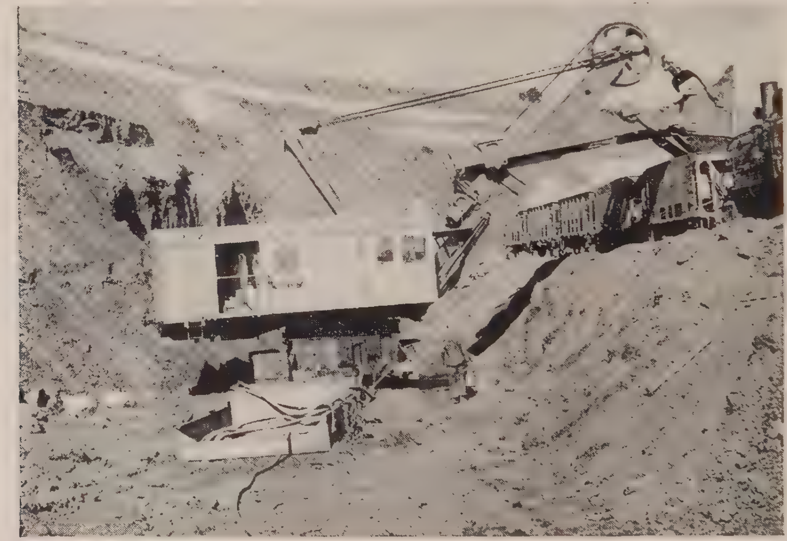
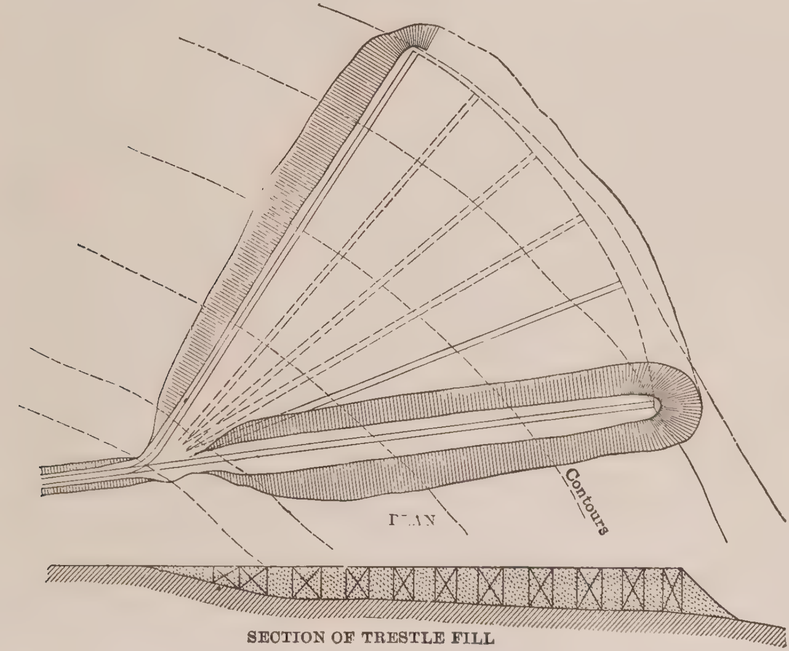


FIG. 199.—A 5-yd. shovel loading ore in iron mine. Note trailing power cable. (*Bucyrus-Erie Co.*)

Ore deposits vary in grade, and sampling indicates high and low grade areas. In pit operation efforts are made to maintain a uniform grade; to do this, shovels are moved and ore is taken from several benches in suitable proportions. Waste rock is often included, especially in the upper parts of the ore bodies, and provision must be made to take such waste rock out to disposal dumps. Sometimes waste rock can be backfilled in the pit when pit bottoms are reached.

Overburden Operations.—Mining overburden follows much the same routine as ore mining. If it is great in quantity, both equipment and operations are engineered to obtain lowest costs. Waste-disposal tracks are laid out on level runs wherever possible to the waste dumps. Bench heights are established in accordance with the nature of the material.

Adverse grades are kept to a minimum. Waste dumps (Fig. 200) may be established on hillsides and afterward are built out by shifting dump tracks from time to time. Spreaders and bulldozers are used to maintain safe edges, together with side-dump cars of 20 to 30 yd. capacity, although in small operations 4- and 5-yd. cars are often used. Waste dumps established upon flat or moderate slopes are started by tracks supported upon trestles at such heights as to give adequate dumping space. After such a dump is started, the pile is fanned out by shifting the tracks. Where trucks are used for transportation, roads to dump areas are prepared and maintained with adverse grades kept to a minimum.



SECTION OF TRESTLE FILL

FIG. 200.—Waste dump on a flat site.

Broken material assumes a slope of about 35 deg. There is a distinct sorting action; large pieces roll to the bottom of the slope and fine material remains in the upper portions. Top edge areas are tender and may settle suddenly and unevenly, so that tracks must be kept away from extreme outer areas. Side-dump cars are discharged upon the top of the pile, while spreaders or dump plows push the spoil over the edge and clear the dumping space. Since waste dumps settle, much track work may be necessary.

Drainage.—Natural drainage contiguous to the pit is by-passed by ditch or flume, and benches are ditched where topographical advantages occur. Intersection of the water table by the lower reaches of a pit requires pump drainage. Sumps are constructed upon the lowest floor of the pit, and pumps and pipes are installed. In some cases drainage shafts are sunk away from the pit and connected to it by crosscuts and drifts beneath the lowest ore level. Pumping units are installed in the shaft. Centrifugals or deep-well pumps, electrically operated, are the usual equipment, reserve capacity being installed to take care of seasonal variation in water inflow. Churn-drill holes connect benches with drifts. Drainage adits are constructed where topography is favorable.

Costs of Open-pit Mining.—The removal of overburden in large amounts imposes a toll upon every ton of ore or mineral exposed by its removal. In one tabulation of open-pit iron mines, ratios of overburden to ore are given over a range from 0.2 to 1.75 cu. yd. per ton of ore; in copper pits the range is from less than 1 ton of stripping up to 6 tons per ton of ore; at Morenci the ratio is 1 to 1 with an ore grade of 1.06 per cent copper. The cost of stripping accumulates until ore can be mined. Much of the equipment and accessories for operation involves large capital expenditures at the start of operations.

At Morenci the sale of $20,000,000 in debentures provided initial capital for starting a 25,000-ton-per-day mining campaign on a 230,000,000-ton ore body containing some 2,500,000 tons of copper. The overburden was estimated at 230,000,000 tons. Initial capital approximated 9 cents per ton of ore and $8 per ton of anticipated copper; part of the capital went for power plant, roads, railroads, mine buildings, smelter, and housing. From financing in 1937 to the first milling operations in January, 1942, about 4½ years elapsed, during which interest upon debentures reached $3,150,000. Time and money are thus required for the initiation of open-pit operations, whether large or small.

Operating costs are another story. In most pit operations labor output in tons per man-shift is large; in iron-ore pits it ranges from 25 to 100 tons per man-shift for ore mining, and in copper pits from 32 to 58 tons. These figures are for large operations; in small-scale pits much lower tonnage per man-shift may be expected. At Sulphur Banks quicksilver mine the output was 4.2 tons per man-shift.

Power requirements in large pits are illustrated by the example of Utah Copper, where in October, 1929, stripping operations required 1.45 kw.-hr. per cu. yd., ore mining 0.79 kw.-hr., and a total of 1.84 kw.-hr. per ton of ore. Direct costs in 1928 at Utah Copper are given by A. Soderberg in respect to stripping: Drilling and blasting 6 cents, loading 4.16 cents, transportation 7.45 cents, track work 7.1 cents, miscellaneous 2.7 cents, and total 27.4 cents per cubic yard. For ore mining the respective figures per ton were 2.72 cents, 1.9 cents, 3.7 cents, 1.7 cents, 1.8 cents, and total 11.7 cents per ton. At New Cornelia direct mining costs were 21 cents per ton (1930), at Chino 21 cents (1929), and at the United Verde 30.3 cents (1928); costs cover only drilling and blasting, loading, and transportation. Total mining costs were 41.2 cents per ton at Utah Copper (1929); 31.4 cents at Chino, and 24.2 cents at Copper Flat; costs include stripping charges. Average open-pit iron-ore mining costs in Minnesota are given by J. J. Craig as follows: Development 27.4 cents, mining labor 12.4 cents, supplies 1.28 cents, miscellaneous 15.9 cents, total 41.1 cents, to which is added a royalty charge of 44.9 cents. The grand total was $1.134 per ton for the period 1926–35. Mining and development averaged 68.5 cents per long ton.

Direct costs per ton for coal stripping are as follows. Montana: Overburden 30 ft., coal 24 ft., total cost of mining and stripping 88 cents per ton; overburden blasted. Illinois: Overburden 30 to 60 ft., coal 5 to 10 ft., total mining and stripping $1.15. Alabama: Overburden 16 to 26 ft., coal 3½ ft.; total cost from 89 cents to $1.54 per ton of coal.

Small-scale operations entail higher costs. At Weepah, Nev., with gasoline-shovel loading and truck transportation over short distances, 300 tons per day were mined at a direct cost of 18.7 cents per ton; stripping cost was 31 cents per ton; total about 50 cents per ton. W. Bradley gives costs of mining 19,436 tons of ore at Sulphur Bank mine in 1936 as $2.24 per ton, including waste removal. At Sloan, Nev., a 300-ton-per-day output of a small limestone quarry gave 10.8 tons per man-shift and a total cost of 48.2 cents per ton for drilling, blasting, hand loading, and tramming.

Cost analysis of open-pit mining is relatively simple. Pit operations are drilling, blasting, loading, transportation, track shifting, and waste-dump distribution. Accessories are tool sharpening, shop repairs, maintenance of electrical lines and machinery, and transport of supplies. Power and supplies, as well as new equipment and repair parts, are purchased in amounts consistent with the scale of operations. Supervision, engineering and clerical services, and managerial direction constitute an over-all charge on operations. Labor pay roll, wages, and labor efficiency obviously are important factors in cost control. Efficient use of power is another factor.

EXAMPLES OF OPEN-PIT MINING

Iron Mining.—The following is summarized from A. H. Hubbell's account of the Susquehanna pit on the Mesabi. This pit was originally operated by track approaches and steam-locomotive haulage but became too deep for economical operation by this method. It was subsequently arranged for shaft operation with a track system on benches and all-electric equipment. A 350-ft. five-compartment shaft was sunk alongside the pit, with the first haulage tunnel 260 ft. below the shaft collar. A second haulage level is 60 ft. below and extends under the pit for drainage and subsequent opening out of a new pit level. A still lower level, 60 ft. beneath, is expected to bottom the ore body. Three 4-yd. electric shovels, Ward-Leonard control, are used for loading. Pit cars and skips are of 4 yd. capacity. Haulage is by differential train. Trains consist of six 4-yd. cars hauled by a locomotive car (a larry), which also carries a 4-yd. load (8 tons); loaded seven-car trains weigh 18 per cent less and empty trains 10 per cent less than corresponding locomotive-drawn equipment. Two 95-hp., 600-volt d-c motors are used upon the larry-car trucks, which have a wheel base of 11 ft. Provision is made for dumping two cars at a time and for hoisting in balance at the rate of one skip (8 tons) per minute. At the shaft collar provision is made for handling three grades of ore and loading 180 railroad cars per day. A remote-controlled larry car delivers the ore from the surface skip pocket to the bins.

The conveyor installation at the Spruce mine has been described a number of times. This mine is also on the Mesabi and was worked with track equipment but was changed over to conveyor operation. Three 10- by 10-ft. vertical transfers connect the pit with a conveyor line beneath the lowest ore level, and ore is delivered to a 500-ton shipping pocket a short distance away from the crest of the pit. The conveyor system is in nine sections having a total length of 4,481 ft. and a vertical lift of 386 ft.; belts are driven at a speed of 500 ft. per min. with a probable maximum of 750 tons per hr. Belts are driven by 75-hp. motors with sequence as well as holdback controls.

At the top of two of the vertical transfers are 100-ft. steel towers set upon circular tracks so as to permit swinging through 360 deg. Anchor towers are mounted on tread tractors, ballasted by 20-ton weights and propelled by 15-hp. motors. A track cable supported by the towers carries a scraper-bucket rig, the scraper cables being operated by winches with Ward-Leonard control. The average radius of operation is 175 ft. but for cleanup purposes can be extended to 750 ft. The 3-yd. bucket used here weighs 3,200 lb. It is of the bottomless, crescent-shaped type, back-tilting when loaded. Buckets discharge upon an 8-ft. grizzly with 16-in. openings; undersize discharges go upon a pan conveyor, which in turn discharges upon a grizzly, oversize going to a crusher and undersize to the transfer. One transfer is equipped for truck dumping; it is likewise provided with a receiving hopper, pan-type feeder, grizzly, and crusher. Each transfer bottom terminates in a hopper serving a pan-type feeder, which discharges upon the belt conveyor in an offset drift. Ore is mined by scraper buckets and in other parts of the pit by a 2¼-yd. electric shovel that loads four rear-dumping trucks, each of 20 tons capacity.

Copper Open Pit.—The following summary of the Morenci open pit is taken from articles by W. C. Lawson and S. F. French. The pit is a mountainside; the layout is similar to that of the Utah Copper, which was also started upon a mountainside. The thickness of the ore body is 850 ft., with an average thickness of overburden of 216 ft.; the final excavation is estimated to cover 350 acres. Pit benches are at 50-ft. intervals with the topmost bench at 5,500 ft. altitude and the lowest at 4,200 ft., an ultimate vertical range of 1,300 ft. Entrance tracks are at 4,600 ft., the coarse crusher station at 4,485 ft.; switchbacks begin at 4,600 ft.

Waste haulage is over a series of tracks laid on a 0.25 per cent grade extending in a direction opposite to that of that of ore movement and serving five waste dumps at 100-ft. vertical intervals, each dump serving two benches by a connecting 3½ per cent grade. Average waste haul is 2½ miles; average ore haul is about 3 miles with 85 per cent of ore tonnage on favorable grade. A series of switchbacks on a 4 per cent compensated grade for 2 miles and then on a 0.4 per cent grade for 1.33 miles brings the ore trains to the crusher plant. Curvature on switchbacks is limited to 10 deg. and on main line to 6 deg.

Standard railroad equipment is used, with main-line tracks of 131-lb. and bench and dump tracks of 90-lb. steel. Shovels, bulldozers, and 22½-yd. trucks were used in the development of benches, roadbeds, and roads. Nine 4½-yd. full-revolving Ward-Leonard-control electric shovels are used for pit operation. Churn drills are electric, with 9-in. bit equipment. Locomotives include six 125-ton diesel electrics, one 100-ton diesel electric, and nine 125-ton trolley-battery units. Maximum daily tonnage of 25,000 tons was planned for three shovels in ore and maximum shift capacity of 6,050 tons per shovel.

TABLE 92.—OPEN-PIT COPPER-MINE DIMENSIONS, IN FEET

Mine	Length	Width	Vertical range	Capping thickness	Ore thickness	Ratio	Over-all angle, deg.	No. of benches
Utah Copper	6,000	4,000	2,000	115	556	1–2	26	24
Copper Flat	4,400	1,500	.....	190	224	....	40–45	11
New Cornelia	3,500	2,000	425	.....	....	0.83–1	45	4
Chino	4,700	3,750	.....	0–150	300–600	1–1	45	8
Sacramento Hill	1,400	1,200	425	280	227	2–1	45	3
United Verde	1,000	800	400	.....	....	3.6–1	45	5
Chuquicamata	7,872	3,280	.....	.....	....	0.4–1	.....	16
Morenci	5,800	4,000	1,300	216	850	1–1	45	26
Flin Flon	1,500	300	280	.....	....	....	60–70	5

Train speeds are 8 m.p.h. on benches and switchbacks, 12 m.p.h. on fair track and 4 per cent grades, 20 m.p.h. on fast tracks from crusher yard at 4,480 ft. elevation to 4,500 ft., and 9.5 m.p.h. on downgrades. Estimates indicated that 30 trains of eleven 80-ton cars per 16-hr. day would be required for the tonnage. Loading time was figured as 6.4 min. per 80-ton car, unloading time as 1.4 min. Side-dump cars are used for both waste and ore. The Morenci pit is an example of hard-rock open-pit mining with the greater proportion of the equipment electrically operated.

In Table 92 over-all dimensions are given for a number of open-pit copper mines. Figure 201 is a photograph of the Chuquicamata open pit and shows the layout of the ore benches.

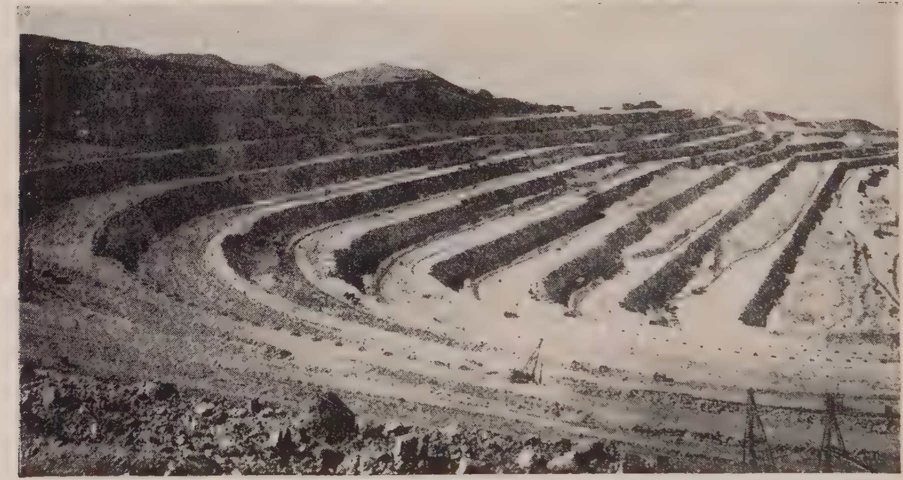


FIG. 201.—Chuquicamata open pit. (*J. B. Huttl.*)

Pebble Phosphate Mining.—C. A. Fulton's description of pebble phosphate mining in Polk Co., Fla., is the source of the following summary. Overburden consists of sand, clay, and thin layers of sandstone. It is removed by an 8-yd. electric dragline, tread-tractor mounted, with Ward-Leonard control and a boom length of 138 ft. The overburden thickness is from 20 to 25 ft.; cuts are 210 ft. in width; moves are 60 ft. requiring 5 min. to move and three 8-hr. shifts to work out the area covered by each move. In making the pioneer cut, spoil is temporarily placed upon unmined ground to be rehandled in subsequent cuts.

The plan of the pit (Fig. 202) is simple. It is divided into four quadrants symmetrically placed about the washing plant, which is placed upon a wide strip or corridor between each pair of quadrants. This corridor provides sump space for the washer. The pioneer cut is started close to the washer alongside the corridor and retreats parallel with the corridor to the property limits. The second cut is at right angles and retreats away from the first cut; on its completion, the dragline moves along the third cut back to the edge of the pioneer cut, a distance of one-half mile, requiring some 5 hr. for the move. The third cut is then drawn back alongside the second, the overburden excavated being cast into the mined area of the second cut. Subsequent cuts are parallel and all start at the same end. Before each move backward the dragline digs 30 to 40 per cent of the matrix from the spoil-bank side, depositing it upon the exposed matrix bed.

Hydraulic mining of the exposed matrix follows closely upon the retreating dragline, the distance between overburden bank and matrix bank (8.5 ft. high) ranging from 200 to 400 ft. Two giants (2-in. nozzles, 360-deg. swing horizontal and 24 deg. up and down vertical) begin excavating at points 25 ft. away from the matrix face. They are supplied by two 8-in. pipe lines stemming from a 12-in. line extending from a pumping station. Each stream is 2,000 gal. per min., of about 200 lb. pressure. The confined streams wash out about 4 cu. yd. of matrix per minute into shallow ditches leading to a sump. At the sump is a pit car or steel sled, on which is a 12-in. sand pump direct-connected to a 300-hp. variable-speed 2,300-volt a-c motor, together with electrical controls. Suction is provided by a 12-in. flexible pipe supported by tripod and rope blocks and fall. Discharge is by a 12-in. pipe line or rock line.

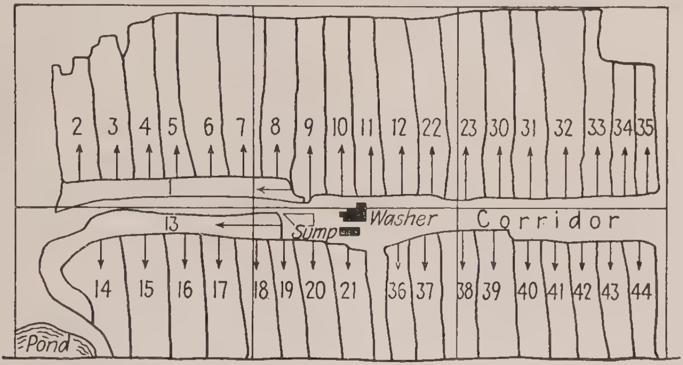


FIG. 202.—Florida pebble phosphate open-pit plan. (*C. A. Fulton, A.I.M.E.*)

When the face of the matrix recedes to a line 125 to 175 ft. from the giants, about every two days, they are moved up; power and pipe lines are extended, the pit car is shifted, and new ditches and sump are blasted out while dragline operations continue. The dragline operator by means of a special cable pulls the giants and pit car to the new position. The move takes from 1 to 1½ hr. The pit delivers 225 to 250 cu. yd. (place measure) of matrix per hour, depending upon whether it is compact or soft clay. Phosphate matrix is abrasive, so that the life of cast-iron pump shells is limited to about 350,000 cu. yd. The rock line delivers the matrix in water suspension to the washer. Booster pumps similar to the pit car are placed at 1,500-ft. intervals between the mining cut and the washer.

Hydraulic mining and sand pumping to washer cost 12 cents per cu. yd. in 1934, during which year 4,413 hr. of operation out of a possible 8,760 produced 1,024,999 cu. yd. of matrix and pumped an average distance of 3,000 ft. The average matrix thickness was 8.46 ft.; power was 5.7 kw.-hr. per cu. yd. Stripping cost was 1.83 cents per cubic yard and stripping power was 0.555 kw.-hr. per cu. yd. of stripping.

Clay Pit.—The Lincoln, Calif., pit of the Stockton Firebrick Company when first visited was 1,600 ft. in length; the vertical range was 75 ft., of which overburden averaged 35 ft. with a maximum thickness of 38 ft. The pit is entered at one end on a 2 per cent 36-in.-gage track. Two beds of clay are mined, 5 and 10 ft. in thickness. A 1½-yd. gasoline shovel is used for handling both clay and overburden. The overburden, andesitic breccia, is drilled by hand-held auger drills and blasted in two benches 50 ft. ahead of the clay bench. Several trains of five 3½-yd. rocker-type cars are handled by two 8-ton Porter gasoline locomotives. The output was 180 cars, with a maximum of 200 cars and a minimum of 150 cars; 12 men handled 300 tons per shift. In 1941 the pit produced 48,000 tons of clay, and 96,000 tons of overburden was removed at a cost of 34 cents per ton.

Limestone.—Limestone deposits are mined by open-pit methods where overburden is not excessive. Thin deposits are mined in a single bench following the cleaning off of the overburden. Thicker deposits are mined in several benches. However, there is a tendency to mine in one bench where practicable. Drilling is by hand-held drills or more often by wagon drills where holes greater than 15 to 20 ft. in depth are necessary. The churn drill is used in higher benches; hole depth ranges from 30 to 100 ft. or more, depending upon the height of the benches. Secondary blasting in varying amount is necessary. Output is 1,000 to 5,000 tons per day in the larger quarries. Loading is by power shovel and transportation by locomotive and train. Trucks are used in the smaller operations. Breaking, loading, and transportation costs range from 15 to 50 cents per ton. Transportation distances are usually nominal, as the quarry product is delivered to a crushing plant or near-by cement mill.

Miscellaneous Minerals.—Quicksilver is mined at Sulphur Banks (California) by open pit. The equipment consists of two 1¼-yd. diesel shovels (Fig. 203). Seven trucks were used, four 6-ton, two 4-ton, and one 10-ton diesel, other trucks being gasoline-operated. A two-stage compressor of 508 cu. ft. capacity and two hand-held drills comprised the drilling equipment. Each shovel loaded an average of 35 tons of ore and 187 tons of waste rock per day.

Coal Strip Pits.—Shallow, flat coal beds that are comparatively uniform, not irregular or broken, are mined by open pits where the overburden is not too thick. The coal must be of commercial grade and markets must be reasonably near. Overburden consists of soil, sand, clay, boulders, shale, and limestone, the seam resting upon fire clay, shale, or limestone. The topography must be level or gently rolling. Preliminary work to determine position, thickness, and coal quality in a given area is by hand churn drills, by boring, or by test pits.

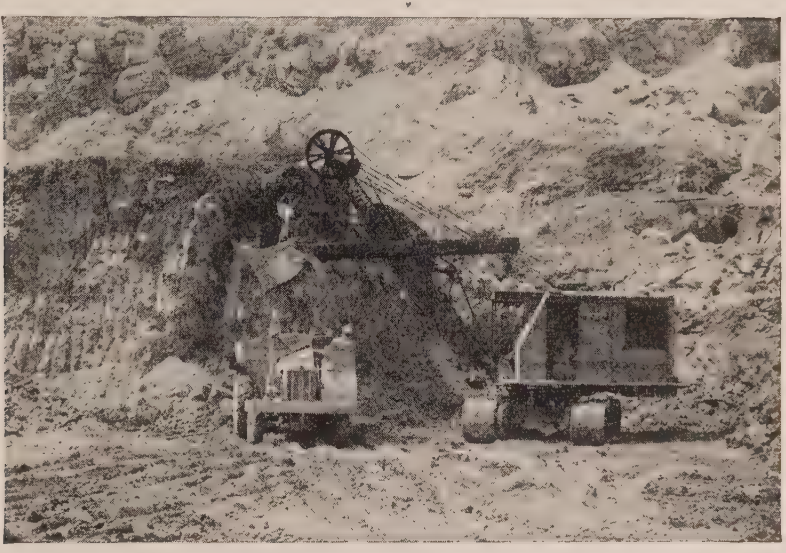


FIG. 203.—Diesel-powered shovel and truck in small pit. (*Bucyrus-Erie Co.*)

The line of the outcrop is the initial point of attack. Overburden is removed in successive strips, 30 to 50 ft. or more in width, and is backfilled into the area from which the coal has been removed. The rate of removal of overburden is such as to uncover sufficient coal to provide the output, which may range from 500 to 6,000 tons per day in individual operations. The ratio of overburden to coal varies in accordance with economic conditions; 10–1 may be taken as a limiting figure, although in some localities it may reach 20–1 or more. Thickness of coal ranges from 18 in. up to 60 to 72 in., in a few cases to 10 ft. or even greater; thinner seams require a greater rate of uncovering and hence higher stripping capacity to maintain an average daily tonnage.

Compact shales and limestones require to be drilled, blasted, and properly fragmented for handling by shovel equipment. Drilling and blasting operations must be balanced with overburden removal and coal area uncovered. Drill-hole ratios are from 10 to 40 cu. yd. per foot of drill hole, and blasting ratios are from 4 to 35 cu. yd. per pound of explosive. Drill holes cost from 6 to 25 cents per foot. Tread-tractor churn drills are used for vertical holes from 6 to 8 in. in diameter that reach to within a few feet of the coal seam. Side-wall drillers (electric or gasoline-powered) are used for horizontal holes placed above the coal in a soft seam; horizontal holes are from 50 to 75 ft. in depth. Where 2- and 4-in. holes are drilled they are usually chambered. Explosives include various grades of dynamite, with lox used in some pits.

Pit equipment consists of drilling machines, strippers, large power shovels (15- to 35-cu. yd. buckets and a large radius of action), smaller power shovels (1.5 to 7 yd.) for loading out coal, and trucks or locomotives, cars, and tracks for transportation to tipple or washer. Dragline excavators are used for stripping overburden that requires no blasting. These follow the stripper (Fig. 204), operating on the top of the same bench and removing the top overburden, overcasting it on the spoil bank beyond the spoil bank of the shovel. Horizontal-thrust coal loaders are sometimes used in place of the power shovel. All equipment is mounted on tread tractors and electrically operated, except for some steam-operated equipment used in older pits. In addition, some equipment is needed for cleaning the surface of the coal before it is loaded. This consists of steel brooms, sweepers, and scrapers of various kinds.

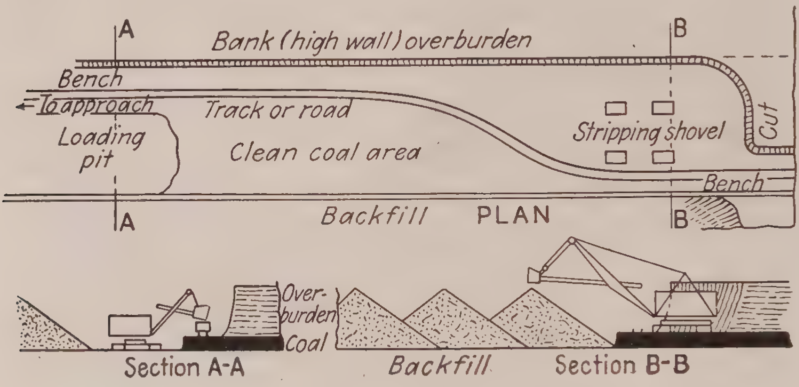


FIG. 204.—Plan and sections of coal strip pit.

The usual method of operation is to remove a 30- to 50-ft. width of overburden from the coal, followed by a similar width of coal offset so as to leave a 25-ft. berm for through tracks or roads. Trucks operate upon the shelf or berm. Tracks, where used, are placed close to the edge of the berm at the face being loaded. As the coal face advances, tracks are shifted from their former position alongside the coal strip that has been loaded out.

Another method of operation places the coal-loading track on the top of the high wall, some 10 ft. from its edge. The stripping cut is 60 to 70 ft. or more in width and the coal cut is the same, extending from the toe of the high wall to the edge of the spoil bank. The coal is loaded into a skip, which is hoisted by a derrick or bank loader and swung over the cars into which its contents are dumped. The bank loader is mounted on a tread tractor; the boom is 70 to 90 ft. in length, and skip capacity ranges from 4 to 10 tons. The track on the high wall is shifted for each strip a distance equal to the width of a strip. A favorable topography is necessary for this arrangement.

Pit layouts and approaches vary in accordance with topography, special conditions, and the location of the tipple or washer. The high wall may be straight or sinuous as contours are followed. In one layout the main road is outside and parallel to the outcrop. Laterals at distances 600 to 1,400 ft. apart extend through the spoil banks and connect with the berm along which traffic flows. The laterals are extended as successive cuts are taken; one lateral may be used for return of empties and another for loads, making a one-way traffic or a run-around. Another layout has the main road beyond the extreme limit of stripping and parallel with the stripping line; laterals 1,000 to 1,500 ft. apart are graded through the high wall or come in at the back door, becoming shorter as successive cuts are made. Still another layout is a single lateral at one end of the stripping limit, upon which two-way traffic is maintained; trucks are turned about upon the berm and backed to loading points. Two such laterals, one at either end of the stripping length, is another arrangement.

The present tendency is to use trucks for transportation. Where the haul is short (2 to 3 miles round trip) trucking costs are not excessive, but for long hauls train and track are more economical. Another variant in practice is a combination system, in which pit trucks deliver coal to a central receiving bin from which conveyors elevate the coal to a loading hopper, at which trains are loaded for transportation to the tipple or washer. Double handling of the coal is the chief objection to this method.

In January, 1928, a coal stripper with 15- cu. yd. bucket, boom 120 ft. and dipper handle 82 ft., was announced as an engineering achievement—which it was. It was followed by strippers of 20 to 24 cu. yd. bucket capacity. In 1936 the 30-cu. yd. stripper attracted attention, and late in 1942 the 35-cu. yd. knee-action stripper with Amplidyne control was described (Fig. 205). The 40-cu. yd. stripper is not far off. A study of the diagram, Fig. 206, which shows quantities of overburden for a 50-ft. strip with various bank heights and corresponding coal tonnages uncovered for seams of various widths, will indicate the need for large digging capacity where daily coal tonnages are large. There is an economic reason for these huge strippers. Fast stripping and corresponding coal production reduce overhead costs and enable maximum market demands to be met. The size of transportation units has kept pace with the increase in the pit capacity. Tractor-trailer units, weighing 20 tons and carrying 33 cu. yd. of coal, have been developed.

Stripper capacities vary with bank height, width of cut, and degree of fragmentation. A. L. Toenges and R. L. Anderson, whose paper supplied some of the foregoing information, recorded stripper yardages at different pits, and from these the following have been taken. A 32-102.5-69, electrically operated stripper gave 700,000 cu. yd. per month from a 20-ft. bank 50 ft. in width; a 30–105–72 gave 1,000 cu. yd. per hr. from a 36-ft. bank 40 to 45 ft. wide; a 18–95–64 gave 600 cu. yd. per hr. from a 39-ft. bank 45 ft. wide; and a 20–92–60 gave 24,000 cu. yd. per day from a 38-ft. bank, 70-ft. cut width. Digging time ranges from 63.5 to 85 per cent of actual time. The numbers given, referring to strippers, have the following signification: the first is bucket size in

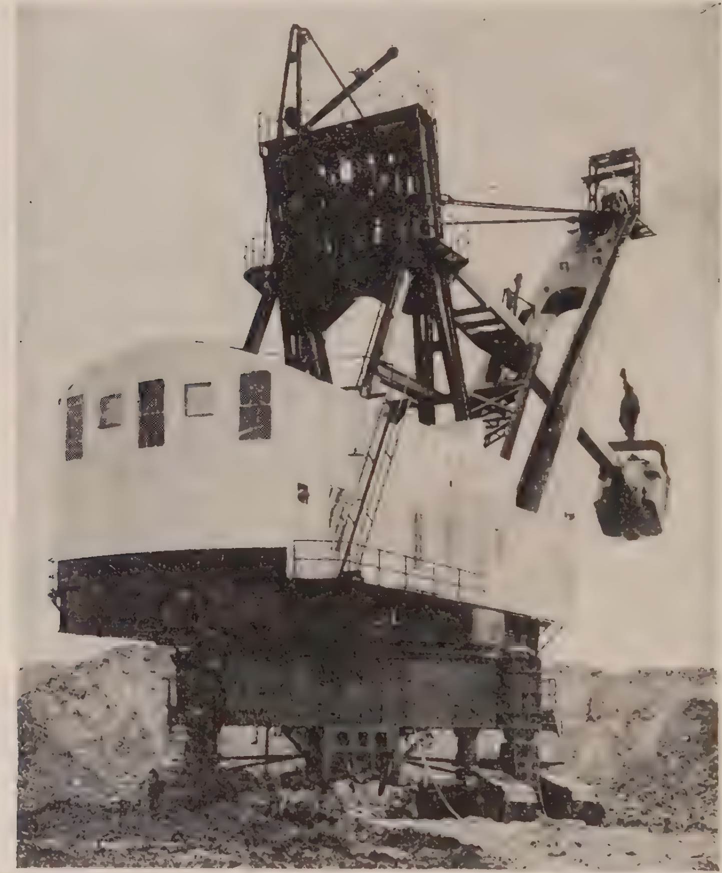


FIG. 205.—Stripper—boom length 105 ft., shovel stick $57\frac{3}{4}$ ft., dipper or bucket 35 cu. yd., cycle time 50 to 55 sec., propel speed 33 ft. per min. (*Marion Steam Shovel Co.*)

cubic yards, the second is boom length, and the third is shovel-stick length, in feet. A 14-cu. yd. dragline with 170-ft. boom gave 225,000 cu. yd. per month. Shovels and draglines are usually operated on three shifts per day. Coal-loading capacities were given as follows: A 1.5-yd. shovel gave 85.5 tons per hr. from a 52-in. seam; a 3.5-yd. shovel gave 257 tons per hr. from a 30- to 60-in. seam; a 5-yd. shovel gave 608 tons per hr. from a 72-in. seam. Tons of coal production ranged from 10.2 to 27.5 per man-shift; man-hours per ton of coal from 0.268 to 0.619. Toenges and Anderson state that a stripping installation cost about $750,000 for shovels and equipment, exclusive of tipple and cleaning plant, as against a cost of $500,000 some 5 years previously. The increased cost of installation represents principally the increased size of the units. Present costs may be expected to keep step with the increased capacity of modern units.

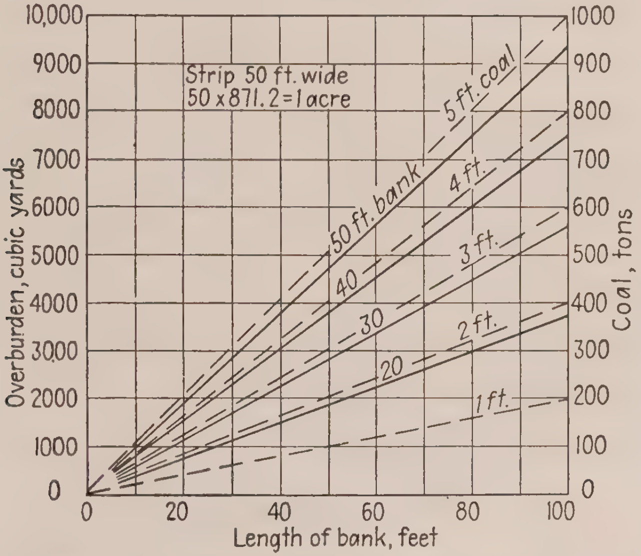


FIG. 206.—Yardage of overburden for various heights of bank, a 50-ft. width and 0 to 100 ft. in length; tons of coal exposed for various thicknesses of coal bed.

Milling System.—Two methods are in use. In one the mill holes are connected by a system of drifts and crosscuts and the excavation can start at the surface of any raise. In the other the drifts are extended from the floor of the pit, no raises being constructed. The loading is effected by the removal of cross boards, which are placed across the lagging supported by the drift sets. The boards at the toe of the loose piles are removed and the ore is allowed to run into the cars in the drift beneath. Breaking is accomplished by drilling and blasting benches. Practically all the material excepting a rib between the drifts can be loaded with a minimum of shoveling. The rib between the drifts must be shoveled. The method has the advantage of requiring the least mechanical equipment for loading. It is effective and economical and is deserving of more extended use in open-pit work where the benches are in excess of 30 or 40 ft. high. Figure 207 illustrates the method, which is in principle the same as underhand stoping.

Glory-hole mining, in which a system of surface "craters" is served by main haulageways and transfers, has superseded tunnel loading to a considerable extent where large tonnage is important. The layout of the method consists of one or more parallel haulageways, from which are offset the chute pockets and transfers as shown in Fig. 208. Transfers, 10 ft. in diameter, are extended to the top of the deposit to be mined by this method, and from the top of the transfers the craters are developed by a series of benches ranging from 10 to 15 ft. in height. Holes are usually more closely spaced than in ordinary bench blasting, in order to secure a greater degree of fragmentation by blasting. Successive benches are opened out from the transfer as the upper benches reach their limit. Oversize is handled by placing a grizzly at the top of the transfer and blockholing oversize pieces at the bottom of the crater, or by offsetting the transfer and establishing a bulldozing chamber and grizzly above the chute pocket. The latter arrangement seems to be preferable. Long transfers, which may reach a vertical height of 300 ft., have the disadvantage of entailing great difficulty when they hang up, and their clearing sometimes involves a considerable risk. To avoid this, R. A.

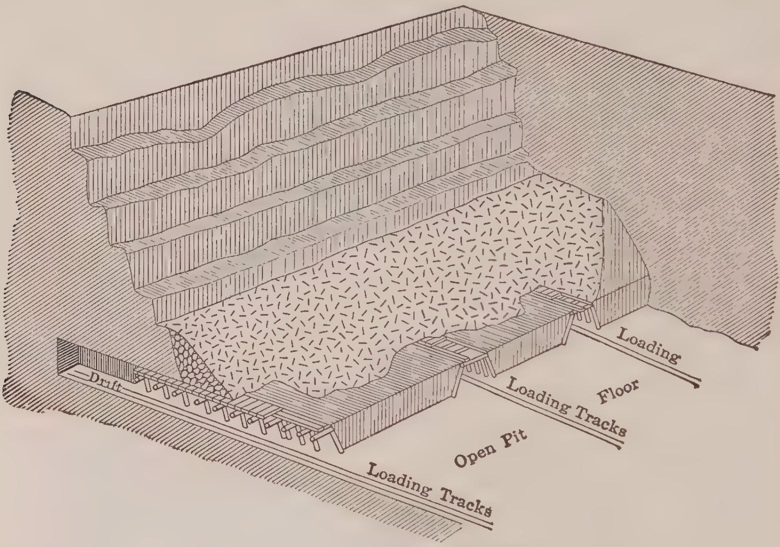
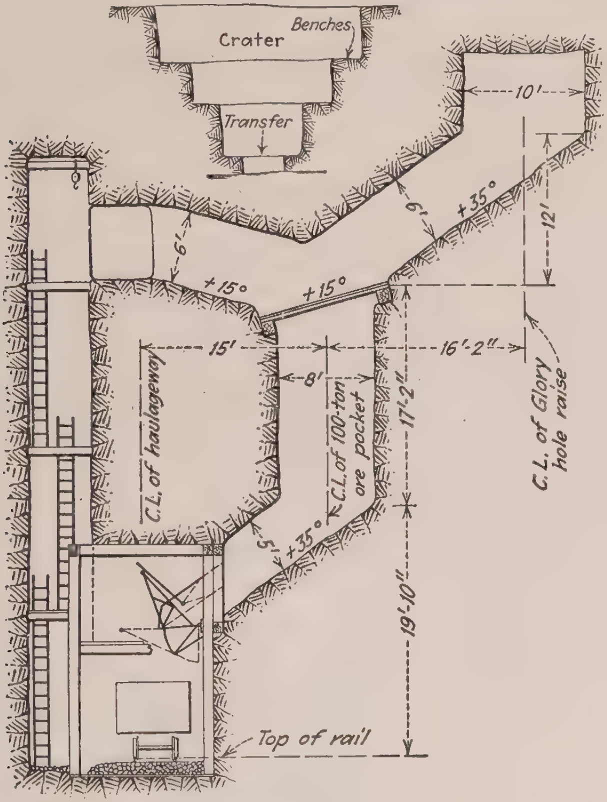


Fig. 207.—Tunnel loading system.

Kinzie makes use of an auxiliary raise from which crosscuts are extended, at vertical intervals of 40 ft., to the transfers. Hang-ups can thus be reached and more easily dislodged than in the long transfer. Where considerable blasting is necessary in bulldozing chambers, either ventilating fans must be used or a system of connecting drifts constructed so as to provide for an air current through the bulldozing chambers.



Side Elevation

FIG. 208.—Cross section of glory hole, bulldozing chamber, and chute.

Chute-gate construction is heavy so as to provide for the handling of large pieces, which sometimes have to be blockholed and blasted at this point. A check gate is also necessary in addition to the chute control. Compressed-air cylinders for moving the chute gate are advisable. A type of successful chute-gate construction is shown in Fig. 208*a*. Grizzly bars are H beams with protecting channels, rigidly held by spacers at several points and supported upon concrete at both ends. Bulldozing chambers are usually somewhat wider than the width of the grizzly, and 1½ to 2 times the length of the grizzly. Height is 8 ft. Connecting drifts are offset so as to deaden the effect of blasting. Provision is also made for safety-rope attachments. As the grizzly-bar spacing is 2 ft. or less, miners are provided with safety belts attached to ropes.

Limestone Mining.—Where overburden is thick or where a particular bed of limestone is to be mined, partial underground mining is practiced. The usual practice is to open out wide entries from the sides of the quarry in the desired bed and then to mine by the room-and-pillar method. There are many variations, from all underground with access by shafts to partial underground and open pit. One interesting example is the quarry near Davenport, Calif., where a limestone deposit of great thickness is mined by glory holes that connect with vertical transfers. These terminate in bulldozing chambers, where secondary blasting takes place. The broken rock is drawn off into loading chutes, where trains are loaded upon a haulage level. Figure 208*b* shows the general layout.

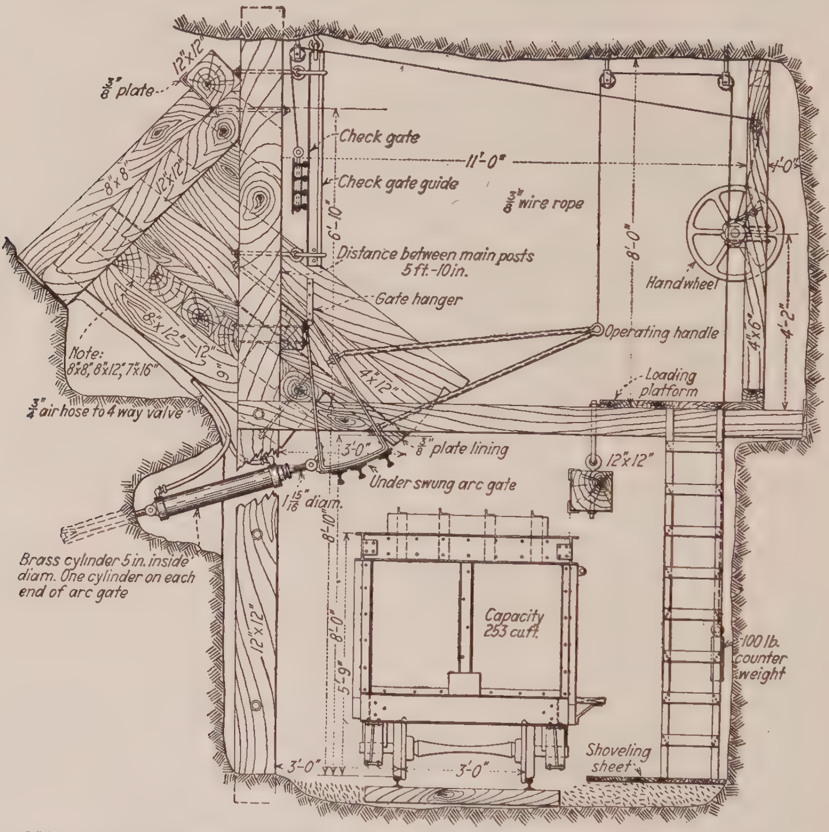


Fig. 208*a*.—Type of heavy chute-gate construction used in glory-hole mining. (*R. A. Kinzie.*)

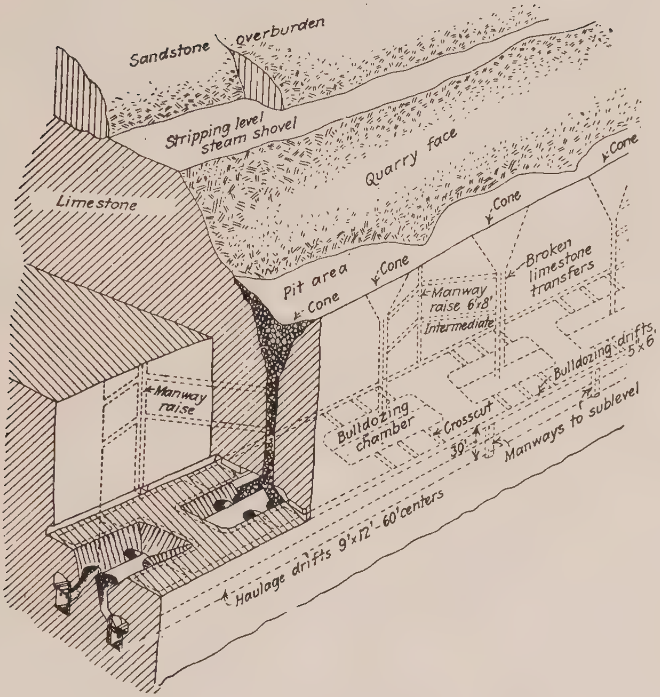


Fig. 208*b*.—Glory-hole mining of limestone at Davenport, Calif.

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CHAPTER XIII

ALLUVIAL MINING

Deposits which are not "in place" and which are detrital or the result of deposition by erosional agencies are mined by special methods. Alluvial or placer deposits may contain gold, tin, platinum, metals of the platinum group, and rare earths. Practically all these substances are characterized by specific gravities higher than those of the accompanying alluvial material, and as a consequence admit of convenient separation by washing in sluices, undercurrents, and upon tables, or by the more primitive methods of pan and rocker.

The methods used in practice are grouped under four heads: Placer mining, hydraulic mining, dredging, and drift mining. The selection of a method presupposes a determination of the controlling features of the deposit. These are

1. The superficial extent or area.
2. The thickness of the deposit; thickness of overburden; thickness of pay gravel.
3. The surface topography and the significant surface features.
4. The slope of the bedrock; its general surface; its physical nature, whether hard or soft, fissured or seamed, rough or irregular.
5. The nature of the gravel, whether fine, medium, or coarse; whether free or compact; whether clayey or sandy; whether large boulders are present or conspicuously absent; whether cemented.
6. The distribution of the mineral; whether uniform, concentrated on or in crevices in bedrock or at intermediate points.
7. The fineness or subdivision of the mineral.
8. The value of the alluvial material.
9. The disposal of the waste.
10. The separation, whether by sluice, washing plant, or jig.
11. The methods available for excavation and transportation.
12. The water supply available and the necessary features required for its development. Ground-water level is important.
13. Transportation, climatic and economic features; taxes and laws.

The most important characteristics of any deposit are quantity, value, and the distribution of value. Shaft sinking and boring are necessary steps to determine these. The determination of the pay streak is important, since operations can sometimes be concentrated and the complete working of the deposit avoided. The distribution of the gold is best studied by comparing sections of the pits or boreholes upon which have been traced curves showing the value distribution in relation to the thickness of the deposit. In Fig. 209 a series of such curves has been drawn that typify some of the conditions met with. In *a* the values are uniformly distributed; in *b* they begin below the overburden and the distribution is approximately uniform; in *c* the values greatly increase as the bedrock is approached; in *d* the values are concentrated at the bedrock, and in *e* the values are concentrated at several depths. With minerals other than gold a somewhat similar distribution may be looked for.

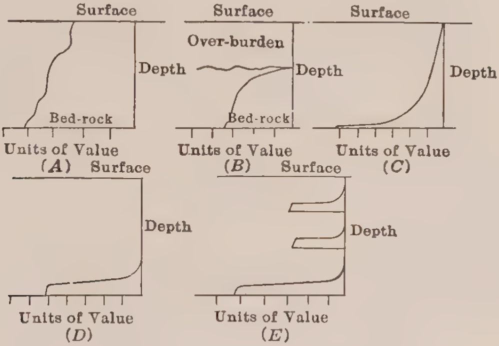


FIG. 209.—Gold distribution curves.

The subdivision of the gold is of importance in determining the details of the washing appliances. Finely divided gold is more difficult to separate than coarse, and the presence of nuggets necessitates the examination and picking of coarse or oversize material. C. F. Hofman gives the following classifications for gold.

NUGGETS:

Coarse gold—that which remains on a 10-mesh screen.
Medium gold—that which passes a 10-mesh screen and remains on a 20-mesh (av. 2,200 colors to 1 oz.).
Fine gold—that which passes a 20-mesh screen and remains on a 40-mesh (av. 12,000 colors to 1 oz.).
Very fine gold—that which passes a 40-mesh screen (av. 40,000 colors to 1 oz.).

FLOUR GOLD:

Purington quotes examples of finely divided gold:
170 colors to 1 c. (314,500 to 1 oz.).
280 colors to 1 c. (436,900 to 1 oz.).
500 colors to 1 c. (885,000 to 1 oz.).

Placer miners often make approximations of value by counting the number of colors obtained from a given volume of gravel and using a factor involving the number of colors to a cent. Purington's estimates of the number of colors to a cent are based upon values of $16 to $18.50 per ounce of placer gold and fine gold valued at $20.67 per ounce.

Minerals other than gold are usually not so finely divided, although as a rule the values in most alluvial deposits are in a finely divided condition and associated with the small alluvial material. This fact renders the use of trommels, grizzlies, and screens for eliminating a part or all of the coarse material a necessity under practically all conditions.

While a certain amount of alluvial mining is done under advantageous conditions in respect to proximity to base of supplies, available roads and railroads, low costs of supplies and labor, and a good climate, much of it is done under pioneering and severe climatic conditions. The alluvial miner more often than not follows closely upon the heels of the explorer and hunter and must adapt his methods to a difficult environment.

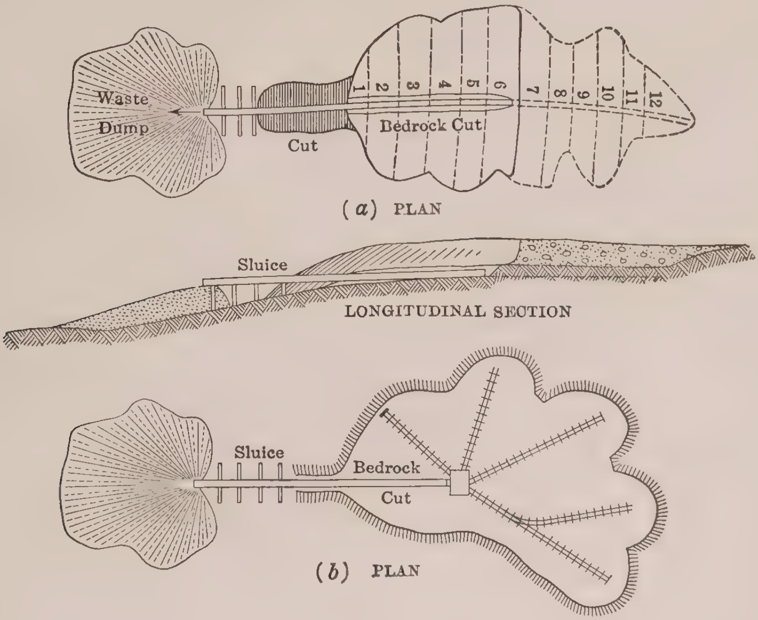
Gold Content per Acre.—In most placer operations the unit taken is the gold content per cubic yard, and the value of the ground is often expressed in cents per cubic yard. The acre is the usual unit of area. The average thickness of the placer deposit and its area will give volume. In Table 93 volume in cubic yards per acre for various thicknesses has been computed as has the gold content in ounces per acre to correspond. Placer ground exceeding 50 cents per cubic yard may be often worked at considerable profit. Good dredging ground may have a value of 10 to 50 cents per cubic yard.

TABLE 93.—GOLD CONTENT PER ACRE

	Oz. per cu. yd.				
	0.02	0.01	0.005	0.0025	
Depth, ft.	Oz. per acre				Cu. yd.
3	96.80	48.40	24.20	12.10	4,840
6	193.60	96.80	48.40	24.20	9,680
9	291.40	145.20	72.60	36.30	14,520
12	387.20	193.60	96.80	48.40	19,360
15	484.00	242.00	121.00	60.50	24,200
Value per cu. yd.	$0.70	$0.35	$0.175	$.0875	@ $35 per oz.
Value per cu. yd.	0.40	0.20	0.10	05	@ 20.67 per oz.

NOTE.—Parts are 1 part of gold per 2,187,499; 4,374,999; 8,749,998; and 17,599,996 parts of alluvial material by weight for the respective ounces per cubic yard.

Manual Methods.—Manual methods of placer operation are restricted to pioneer conditions, small rich deposits, thin scattered deposits, and remote places without roads or convenient access, or where it is impracticable to obtain machinery or to develop water supplies for hydraulic mining. There are two fundamental methods of hand operation, shoveling into sluice boxes and ground sluicing. For the application of the first a water supply is obtained by damming a stream and conducting water by flume or pipe to the area to be worked. A sluice box is constructed in a bedrock cut at the lower end of the deposit; a strip 12 to 15 ft. in width is worked along the axis of the gravel bed, the sluice box being extended as the excavation advances upgrade. The gravel is shoveled into the box and is disintegrated by the water as it flows over the riffles.



(a) PLAN

(b) PLAN

Fig. 210.—Placer mine.

Boulders and large stones are stacked and the bedrock cleaned as the strip is extended; on completion of the initial strip the sluice boxes are shifted parallel with the bank and a new slice is taken. See Fig. 210.

Ground sluicing is started by running a stream of water over the surface and starting a bedrock cut which is extended upstream until the limits of the deposit have been reached. At this point the stream of water is directed against the bank at its upper end and a strip some 20 ft. in width and downstream is worked off. Shear boards and wing dams bring the water along the face. Bedrock cuts direct the flow of the water and gravel to the main bedrock cut, which serves as a sluice, the rough bedrock playing the part of riffles. Sluice boxes may be used and are preferable. Bedrock and cuts are cleaned up from time to time and the material is concentrated by pan and rocker. Boulders and large stones are stacked upon bedrock after it has been cleaned. Booming is resorted to where normal stream flow is insufficient for ground sluicing. In this variation, water is reservoired and then automatically released to flow in sufficient volume along the ground-sluice face. The bedrock must have sufficient grade to give cutting and transporting velocity to the stream flow. In both manual methods, power-driven pumps are sometimes used to supply water to the placer deposit from a near-by stream that may be too low for gravity flow.

Wing-damming is a method used in working river-bed placers. A dam is constructed in the river bed above the ground to be worked. A similar dam is constructed below. Both dams are connected on the river side by a watertight wall. The area thus enclosed is worked out, the pit being drained by a "china pump" or a centrifugal.

Fluming is a method used in river-bed working. The length of river bed is cut off by an upper and lower dam and the water flow is carried between the dams in a wooden flume. Drainage and working follow.

Drainage of pit workings is accomplished by ditches which are constructed on a low grade and tap the bedrock at the lowest point. Deep pits in valley bottoms are protected by damming the stream and deflecting its course around the pit. Where drain ditches are impracticable the "china pump" operated by a water wheel is frequently used. Where this is impracticable a centrifugal pump of the horizontal or vertical type is installed.

Sluices.—The sluice is a washing or separating device. Before satisfactory separation can be accomplished the gravel must be disintegrated. Disintegration is practically effected in excavation and completed in the sluice. The length of the sluice is the important dimension controlling disintegration. Gravels that are easily loosened require a relatively short sluice, while gravels that are compact or contain much clay must pass through a long sluice before all the fine material is broken up and mixed with the water. At the dump box, or receiving end of the sluice, a punched plate screen or grizzly is often placed to receive the gravel. This separates the large stones and coarse gravel, leaving only the fine material to pass through the sluice. A grizzly with 1-in. bars or a punched plate with ½-, ⅜-, or ¼-in. holes is a common arrangement.

The sluice box is constructed in 12-ft. sections butted together and held in place by wooden strips nailed on the outside. Telescoping sections are used where the boxes must be frequently moved. While wood is the customary material used in the construction, steel offers many advantages both in freedom from leakage and ease in handling. The width of the sluice ranges from 10 in. up to 6 ft. and the depth from 1 up to 3 or 4 ft. Riffles are placed in the bottom either transversely or longitudinally. Many different kinds of riffles have been devised by the placer miner. Usually the available material determines the selection. Round wooden poles, poles protected by iron strips, wooden blocks, cobblestones, steel rails, metal grids, and Hungarian riffles are some of the kinds used in sluice boxes where the ordinary run of coarse and fine gravel must be handled. For fine sand and gravel, smaller riffles or carpet, blanket, burlap and coco matting protected by expanded metal or riffles are used. The form of the riffle and the spacing should be such as to permit large stones to slide or roll along the bottom of the sluice. Riffles are subjected to intense wear, and the surface receiving the wear should be protected by steel strips wherever possible. Figure 211 illustrates a number of types of riffles and timber and steel boxes.

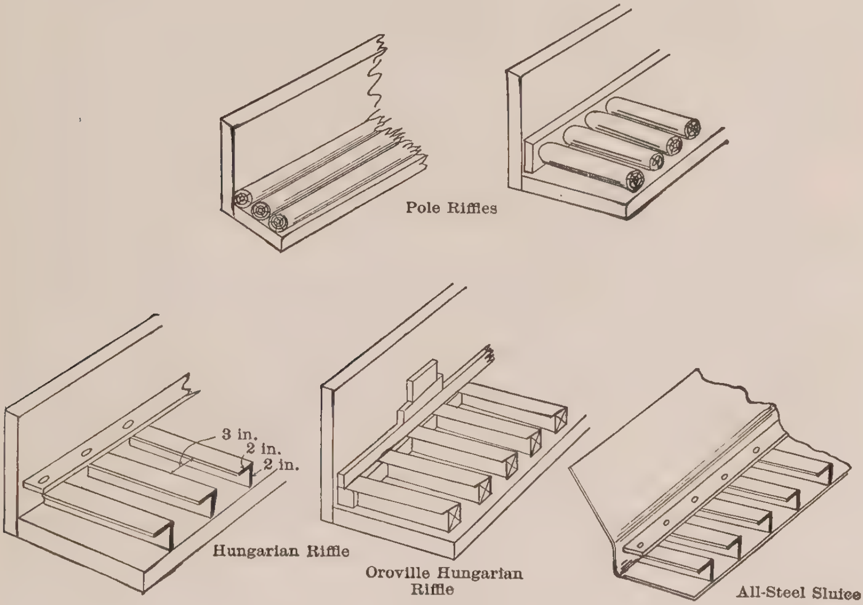


Fig. 211.—Riffles.

The treatment of fine material containing fine gold is best effected by shallow wide sluices set on comparatively steep grades. Coarse material requires a deep, narrow sluice. The grade ranges from 2 to 12.5 per cent. The smaller grade is about the lower limit and is suitable for light sands and fine gravel. The commonest grade is 4.16 per cent, or 6 in. per 12-ft. section. With this grade it is necessary to remove the larger stones by "forking." Steeper grades give an increased duty for a given flow of water.

The width of the sluice and its depth determine its capacity. Two examples of capacity are given in Table 94.

Table 94

Width, in.	Depth of flow, in.	Grade, per cent	Water flow, cu. ft., min.	Cu. yd. gravel, 24 hr.
10–12	6–7	4.16	45	67.5–135
12–14	10	6.2	100	150–300

The *Engineering and Mining Journal* gives the capacities of large sluices in the following table:

Table 95

Width, ft.	Flow, miners' in.	Flow, cu. ft., min.
3	200–600	300–900
4	400–1,200	600–1,800
5	1,000–2,500	1,500–3,750
6	2,000–4,000	3,000–6,000

Note.—Depth of sluice ranges from one-half to three-fourths of the width; depth of water flow one-third to one-half clear inside depth.

The quantity of gravel handled by the sluices ranges from 1.5 to 3 cu. yd. per cu. ft. per minute of flow per 24 hr. Given the grade and water flow, the velocity and cross section of the stream can be calculated from Kutter's formula, with allowance for turbulence of flow.

The length of the sluice will depend upon topographic conditions and upon the nature of the gravel. Short sluices do not give much opportunity for the gold to be caught. With small sluices (12 to 14 in. wide) and with coarse to medium gold, from 36 to 72 ft. may be all that is necessary. With compact gravel and the other conditions the same, from 200 to 300 ft. may be necessary. One authority states that the sluices should not be less than 240 ft. long.

Manual Methods Plus Mechanical Aids.—The placer miner has tried many mechanical aids to facilitate the working of deposits larger and deeper than can be handled by manual methods. Wheelbarrows, horsedrawn scrapers, and portable track and car are used in the smaller operations (see Figs. 212 and 213). Power-operated derricks and scraper rigs of various kinds have found some use. In Alaska the Bagley scraper has been used for small pits. This scraper is a three-sided box without bottom, with a cutting edge and tail-rope attachment on the end plate. A double-drum steam hoist is used for the main rope and the tail ropes. The tail-rope sheave is anchored to a cable so that its position may be easily changed. Another rig comprises two tail ropes, a three-drum hoist, and two sheave anchorages at the far corners of the pit. The scraper discharges into a car that is hoisted up an incline and automatically discharged into the dump box of the sluice. Another rig employs a ramp up which the loaded scraper is hauled and discharged into the dump box (see Fig. 214).

Cableway rigs have also been successfully used. The track cable is fastened at one end to a strongly guyed mast in such a position as to bring the bucket over the dump box of the sluice; the other end is anchored to a cable so that its position may be varied, thus covering a wider area. A carrier supports the ropes to which the excavating bucket is attached and by which the bucket is drawn along the gravel, elevated, and taken to its dumping position above the dump box. A fan-shaped area can be worked extending out from the dump box to the anchorage limits.

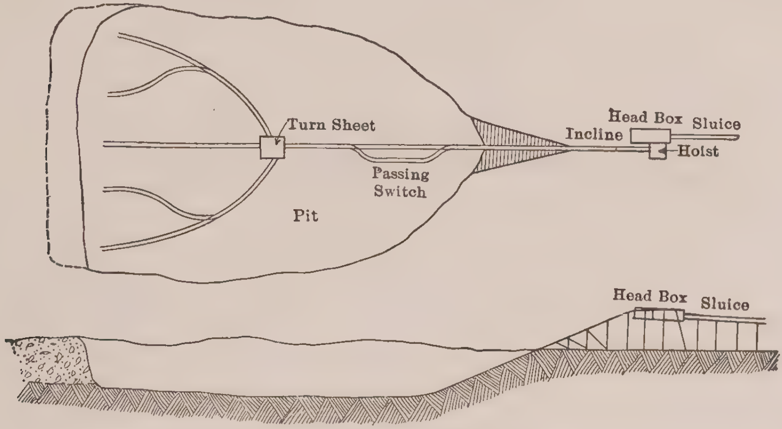


Fig. 212.—Placer mine served by incline.

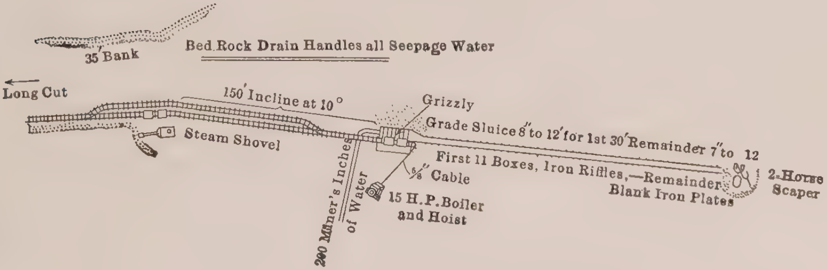

35′ Bank
Bed Rock Drain Handles all Seepage Water
Long Cut
150′ Incline at 10°
Grizzly
Grade Sluice 8″ to 12′ for 1st 30′ Remainder 7″ to 12
Steam Shovel
First 11 Boxes, Iron Riffles,—Remainder Blank Iron Plates
5/8″ Cable
2-Horse Scaper
200 Miner's Inches of Water
15 H.P. Boiler and Hoist

Fig. 213.—Placer mine. (*U.S. Geol. Survey.*)

Dragline excavators have also been used upon larger and thicker deposits. In all arrangements of this kind the problem is excavation and transportation to a height sufficient for a sluice grade with a minimum of labor and with mechanical appliances not too costly or too heavy. The position of the dump box is the focus being elevated to a sufficient height to give grade and dumping area; when the limits of a pit are reached another pit is established and the equipment moved to a new location.

Costs.—Accurate estimates of the cost of placer-mining operations are difficult to obtain. Perhaps the best summary is represented by the work of C. W. Purington in Alaska.[1] Purington's figures represent operations under severe climatic conditions, difficult access, high wages and cost of supplies, and relatively thin deposits of moderate extent. They are of value in establishing the upper limit of cost. From his comparative cost table the data presented in Table 96 have been selected.

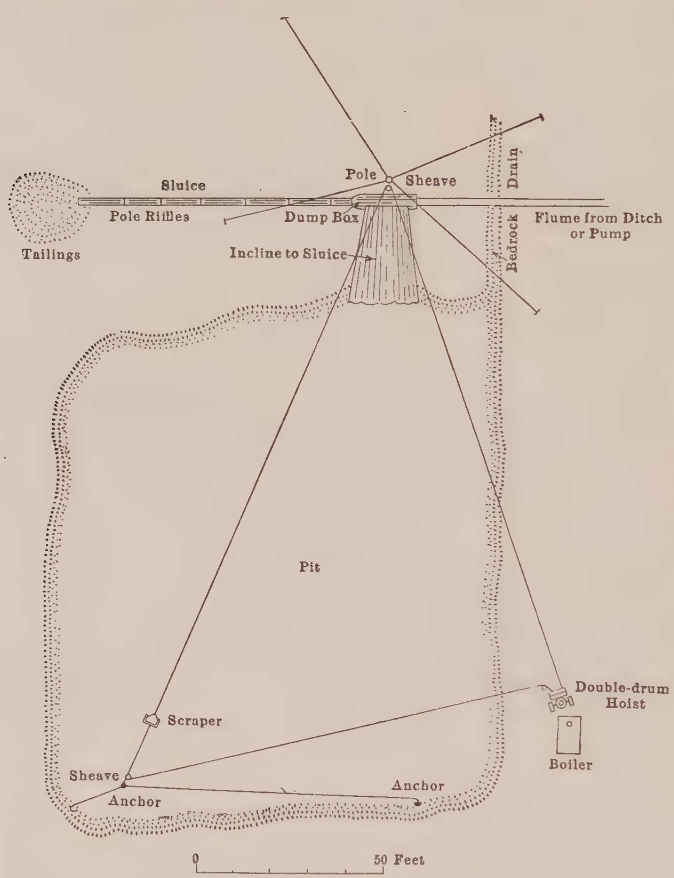


Fig. 214.—Placer mine operated by power scraper. (*U.S. Geol. Survey.*)

The rate for shoveling into sluices in different districts in Alaska as given by Purington is presented in Table 97.

The lowest cost of shoveling into sluices in Alaska is given as $1 per cubic yard, and from that the range extends up to $5 in the interior parts of Alaska. To these figures from 10 cents to $1 must be added for the cost of stripping muck or overburden. The high proportion of labor required in this method of mining is indicated by the above figures and the necessity for mechanical appliances is obvious. The cost of the plant required for the shoveling-in method is given as ranging from $500 to $2,000. The plant includes a seepage water drain, a sod dam, a ditch to sluice, and a string of 10 sluice boxes.

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[1] *Bull.* 263, *U.S. Geol. Survey.*

TABLE 96

	Interior Province				Seward Peninsula			
	Capacity, cu. yd. per 24 hr.	Thickness deposit in ft.	Thickness of gravel worked in ft.	Cost per cu. yd.	Capacity, cu. yd. per 24 hr.	Thickness of deposit in ft.	Thickness of gravel worked in ft.	Cost per cu. yd.
Open cut; shoveling into sluice, including stripping, no pumping.....	63	8.6	3.5	$2.39	145	6.6	3.3	$1.87
Open cut; horse scraping...........	105	20.0	10.0	0.60	200	5.0	5.0	0.46
Open cut; shoveling, wheeling to bucket cable tram to sluice........	162	17.5	4.5	2.14				
Open cut; shoveling into cars, track and incline to sluice.............	450	14.0	5.0	2.43				
Open cut; shoveling into buckets, derricking to sluice...............	233	15.0	9.0	1.75	550	15 0	11.0	0.91
Open cut; steam shovel, track and incline to sluice..................	800	22.0	22.0	1.46	1000	30.0	27.0	0.52
Open cut; steam scraping..........	92	15.0	8.7	0.49				
Booming with self-dumping gate....	250	7.5	6.0	0 07				

NOTE.—Interior Province—Atlin district, B. C., Klondike, Fortymile, Eagle, Birch Creek, Fairbanks, and Rampart districts.

Seward Peninsula—Nome, Council, and Solomon districts.

Wages—Atlin district, $3.50 to $4.50; Fortymile, $5; Fairbanks, $10 (no board): Nome, Council and Solomon, $5 per day. Board is included in the day's wage except where noted.

TABLE 97

Lift, ft.	Thickness of pay gravel, ft.	Rate of shoveling, cu. yd. per 10-hr. day	Remarks
9	2	3.5	2-stage shoveling, Bonanza Creek.
6	4.41	5	Average 12 operations on Birch Creek.
....	....	2.75	Large boulders—American Creek.
....	5	4	Discovery Fork
5	3	7.5	Fairbanks average of three operations.
....	5–7	5.76	Nome.
....	3	9	Anvil Creek.
High lift	3	3.75	Solomon River.
....	3.5	6.63	Seward Peninsula.
5	3	12	

Purington's cost and performance figures are to a considerable extent applicable to present-day conditions after some allowance is made for dollar depreciation. Variation in wages also needs some consideration, but relatively low wages prevailed in Alaska when Purington's observations were made.

Horse scraping into sluices involved an expense of from $17 to $23 per day for a team of horses and driver. The output is from 30 to 40 cu. yd. moved a distance of 75 ft. At Penelope Creek, Seward Peninsula, the cost for this method reached 30 cents per cubic yard. With a breaking plow and team of horses to four scraper teams, costs as low as 25 cents per cubic yard, not including top stripping, were obtained.[1]

The plant required for the steam-scraper method costs $3,500 in the Klondike district and consists of a double-drum steam hoist, 25- to 30-hp. boiler, scraper, ropes, and anchorages. The capacity is given as 250 cu. yd. per 24 hr. and the cost, 49 cents per cubic yard.[1]

The cost of a derricking plant on Pedro Creek, Fairbanks district, and a suspended cable and carrier plant are approximately the same, about $4,500. The capacities are, respectively, 233 and 200 cu. yd. per day. The cost per cubic yard with the derricking plant is given as $1.75.[1]

J. P. Hutchins points out two important facts concerning mechanical equipment: Bulky and heavy machinery should be avoided, and mobility of the equipment is all-important in placer operations. Respecting the latter point, he states that any mechanism that is not easily mobile must have great compensating advantages to make its use advantageous.[2]

General.—The yardage, width, length, and other proportions of placer deposits are given in Table 98.

TABLE 98

Width, ft.	Length, ft.	Area, sq. ft.	Cu. yd. per 3 ft. of depth	Thickness, ft.	Cu. yd. per acre
50	871.2	43,560	4,840	6	9,680
100	435.6	43,560	4,840	12	19,360
200	217.8	43,560	4,840	24	38,720
300	145.2	43,560	4,840	30	48,400
500	87.1	43,560	4,840	60	96,800

TABLE 99

Rate of working, cu. yd. per day	Days required for 1 acre, 3 ft. thick	Acres worked per 120 days
50	100	1.2
100	50	2.4
200	25	4.8
300	17	7.0
400	13	9.2
500	10	12.0

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[1] Purington, reference cited before.

[2] *Bull.* 345, *U.S. Geol. Survey*, p. 71.

The rate of working, time in days required to work 1 acre of ground 3 ft. thick, and acres worked in a working season of 120 days are given in Table 99.

The distribution of plant cost and the yardage worked are given in Table 100.

TABLE 100

Cu. yd. in deposit	Capital in plant and plant cost per cu. yd.			
	$1000	$4000	$5000	$10,000
5,000	0.20	0.80	1.00	2.00
10,000	0.10	0.40	0.50	1.00
20,000	0.05	0.20	0.25	0.50
40,000	0.025	0.10	0.125	0.25

Hydraulic Mining.—Water under a pressure of from 50 to 650 ft. is used in this method for excavating and washing gravel to the sluices, in which it is disintegrated and transported to waste, the gold and heavy minerals being caught by the riffles. A water supply at a suitable elevation above the deposit is developed and the water is carried by ditch, flume, or pipe to the pressure box or penstock supplying the pipe system of the pit or hydraulic mine. Reservoirs may supply sufficient water for continuous operation, but some hydraulic mines start when water is available and close when water supply becomes insufficient.

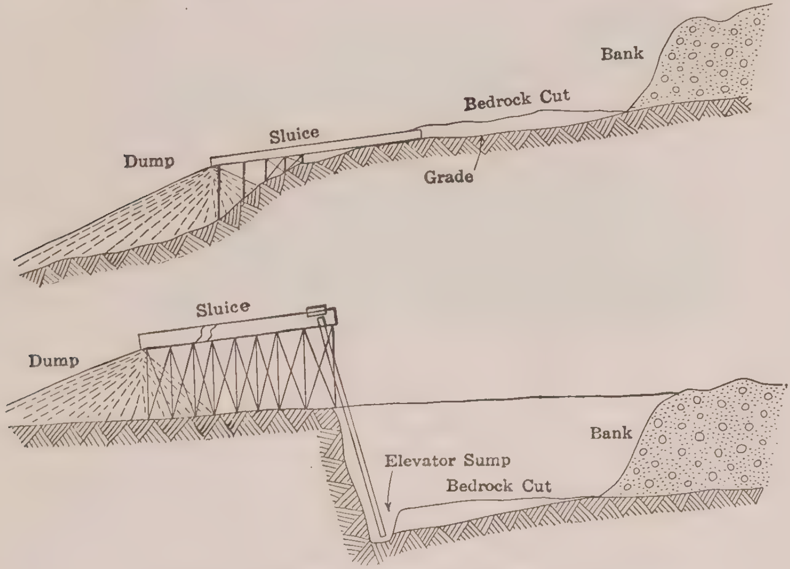


FIG. 215.—Sectional views of hydraulic mines.

For best results, bedrock should be of sluice grade, not less than 1.5 per cent and preferably 4.5 per cent or more (see Fig. 215). The area for waste disposal should be greatly in excess of the area to be mined. Deep bedrock cuts gain additional grade but are costly. In some favorable cases a tunnel can be driven beneath the deposit, and both grade and dumping area can be secured thereby. Some pits admit of development by tunnel and shaft extending through bedrock to the lowest bedrock point of the gravel bed. A topographical survey and accurate measurement of bedrock contours, as well as measurement of the proposed dumping area, are indispensable preliminaries. The determination of values present is made by bores, shafts, drifts, and crosscuts. These also give information about bedrock positions, which is of great importance where sluice tunnels are contemplated.

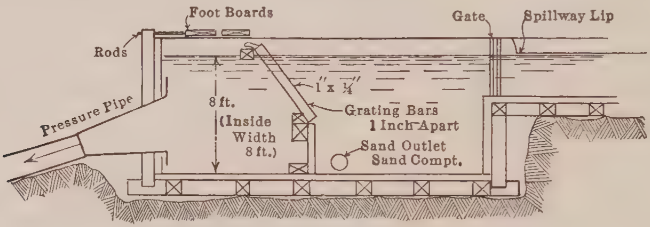


Fig. 216.—Head box.

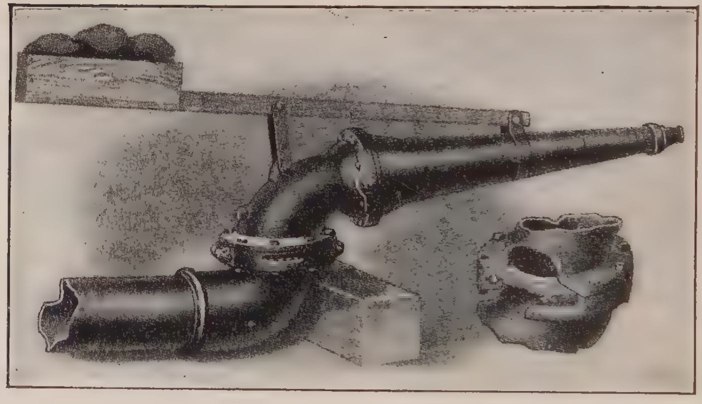


Fig. 217.—Hydraulic giant.

At one time the hydraulic elevator was an important adjunct to hydraulic mining where river beds were worked and where insufficient grade had to be overcome. Figure 215 indicates the type of working wherein the hydraulic elevator afforded advantage.

A head box is shown in Fig. 216. The hydraulic giant is illustrated in Fig. 217. A double joint enables it to be swung in a horizontal plane over the complete circle, and it may be elevated or depressed in a vertical plane. Nozzle sizes range from 2 to 8 in. in diameter. The supply inlet is 2¼ times the nozzle diameter. Its weight ranges from 390 to 2,075 lb. A counterbalance gives better control and the larger sizes are equipped with a deflecting nozzle that enables the pipeman to move the nozzle with but little effort. In Table 101 the water requirement under various heads and for different sizes is given as taken from Campbell's Hydraulic Elevator Catalogue.

TABLE 101

Head, ft.	Theoretical spouting velocity, ft. per sec.	Cu. ft. per sec. and diam. of nozzle (in.)			
		5	6	7	8
100	80.25	10.17	14.64	19.94	26.08
200	113.5	14.34	20.64	28.20	36.80
300	139.0	17.62	25.44	34.54	45.12
400	160.5	20.35	29.28	38.89	52.16
500	179.4	22.75	32.80	44.00	58.24

In the simplest cases operations begin at the lowest part of the deposit. A bedrock cut is made and a sluice is installed; sluices are from 3 ft. wide by 2.5 feet deep up to 6 ft. wide by 4 ft. deep, of sufficient length to reach the dumping site. Pressure water is handled by a hydraulic giant, one or more being employed in the pit. The mining is started by undermining the bank and washing the gravel to the sluice head, where wings direct the flow into the sluice. The height of gravel bank, its tightness or looseness, the grade of sluice, and the available waterhead determine the rate of mining with a given water flow. With high sluice grades and not too flat bedrock, water duties are high. California gravels gave 1 to 4.5 cu. yd. per miner's inch per day (1.5 cu. ft. per min.); with fine material, 5.5 cu. yd. per miner's inch; and with cemented gravels, considerably less. Large boulders and rocks are dislodged and left in the pit or may be stacked by bulldozers; stones up to 15 to 20 in. are transported by the sluice; bedrock is often loosened for a foot or more in thickness and washed through the sluice; hard bedrocks must be hand-creviced and cleaned; cemented gravels must be blasted before washing.

The hydraulic mining sluice serves a triple purpose: It completes disintegration, it transports the water and gravel to the dump, and it catches the gold and heavy minerals in its riffles. Two general types of riffles are in use, one arranged longitudinally in short sections and the other tranversely. Transverse Hungarian riffles facilitate disintegration, and longitudinal riffles help transportation; both are efficient mineral savers (see Fig. 218). Wooden blocks with spacers are used for transverse riffles; and poles, often shod with steel plates or bars, are used for longitudinal riffles. Wooden sluice boxes are used in many instances. In more modern installations they are of abrasion-resisting steel, and riffles are angle iron or steel rails, made up into short sections that can be handled in cleanups. Transverse steel riffles are given a slight upward cant against the stream flow and are spaced 2 or 3 in. apart; longitudinal riffles are given the same spacing. Cobblestone riffles are nearly obsolete.

At the La Grange mine (California), riffle wear was determined for different materials and the results were published in the *Engineering and Mining Journal* of May 24, 1913, as follows: Wooden riffles wore out after flowing some 135,000 to 270,000 miner's in.; rails spaced 5 in. apart and placed longitudinally wore out at 400,000 in.; with the same spacing but placed traversely the wear indicated 600,000 miner's in.; manganese steel spaced at 5 in. and placed transversely showed 2,200,000 miner's in. The manganese steel riffles handled some 12 to 15 million cu. yd. of gravel.

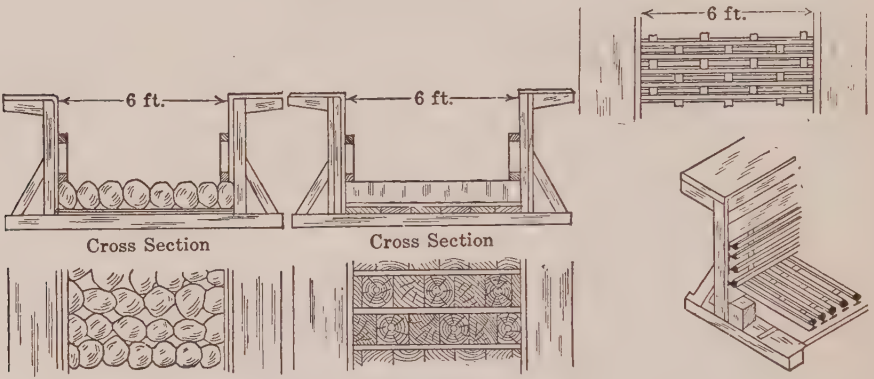


FIG. 218.—Sluice construction.

Distribution of Gold in Sluices.—Particle size of the gold, rather than the length of the sluice, determines the distribution of the gold in the sluice. At the La Grange mine, where the greater part of the gold was caught in the first 250 ft. of the sluice, studies were made of the coarseness of the gold and its distribution. The results are given in Table 102.

Cleanups are made at frequent intervals in the upper sections of a sluice; the lower sections may be cleaned only once a season. To guard against theft, various devices are installed at the head boxes of the sluice and watchmen are employed. Drops are sometimes placed in the sluice line to increase disintegration, and at such points undercurrents with 8 per cent grades are also placed to catch fine gold. At a drop and undercurrent a grizzly or punched plate is placed near the end of the sluice bottom. The fines passing the grizzly are distributed over several wide, shallow tables equipped with Hungarian riffles spaced about an inch apart; the discharge of the tables is collected into a launder and returned into the lower section of the sluice below the drop.

TABLE 102

Box[a] no.	Mesh size and weight of gold						Total, oz.
	+10 oz.	−10 +50 oz.	−50 +100 oz.	−100 +150 oz.	−150 +200 oz.	−200 oz.	
5	45.8	50.7	1.38	0.36	0.31	1.45	100.0
6–16 incl.	18.0	83.30	2.33	1.00	0.31	0.83	105.77
22	1.73	20.22	3.08	0.70	0.25	0.62	26.60
48	0.18	2.18	1.06	0.12	0.05	0.16	3.75
88	0.018	0.12	0.47	0.008	0.026	0.005	0.647
136	None	0.053	0.027	0.043	0.011	0.01	0.144

[a] Box length 12 ft. (total length of sluice 3,600 ft.)

In a tunnel approach, sluice boxes are placed in the tunnel and wings are constructed at its entrance or at the upper terminus of the sluice. Pit operations are much the same as has been described. In the case of tunnel and shaft transfer, the pit is opened out at the mouth of the shaft by carefully working the bank, hydraulicking the gravel into the shaft and to the sluices, which begin at the side of the shaft near the bottom.

Placer deposits of moderate thickness and very low grades are sometimes operated by hydraulic elevator (Fig. 215). A pit is dug and the elevator is installed. The hydraulic jet lifts the gravel and water to a sufficient height to discharge into a sluice set up on bents, high enough to get sluice grade and dumping space. A grizzly is placed at the sump of the elevator and the gravel is washed into the sump, oversize being removed from the grizzly at intervals. The comparatively low (20 per cent) efficiency of the elevator and the fact that such deposits can better be worked by dredging have limited the application of the hydraulic elevator to small deposits and special conditions. The use of a dredge pump (centrifugal) for elevating to sluice boxes from the sump of the pit has been tried out; under favorable conditions it may be a workable arrangement for flat deposits. At one time gravel pumps were used in Malaya to some extent to elevate to sluice lines. The extent of the abrasion of pump and pipe is the uncertain factor, and although abrasion-resisting steels and alloys are used in construction of the pump parts, operating costs may be high and initial efficiencies may be lessened by wear of pump parts. The over-all efficiency of the dredge pump is greater than that of the hydraulic elevator. Diesel power gives a low power cost.

Shallow stream-bed deposits (12 ft. in thickness) and low grades can be handled by setting three or four sluice boxes (a sluice box is usually 12 ft. long) in a bedrock cut at the lower end of the bank, wing-boarding the sluice entrance and piping from the top of the deposit into the sluice from an upstream position. Another giant is placed close to the lower end of the sluice and the tailings are "blown" to the side of the stream. Pits of a length 300 to 450 ft. and to a width of 100 ft. or more have been successfully worked.

Wimmler describes piping over the side of sluice boxes as follows: To start, 8 to 20 boxes are installed in a bedrock trench and wing boards are placed at the sluice entrance; next a deep cut is extended by hydraulicking along the axial line until the upper part of the deposit is reached; sluices are then installed through the length of the cut. Piping starts at the upper end. Parallel angling side cuts pointing downstream are made. The gravel is washed over the side of the sluice as the cuts are made in sequence, one side being piped while the bedrock on the other side is being cleaned up. Sluices are placed so deep in the cut that the sides are below the bedrock surface. The sides of the sluice are protected by slabs or old boards. Wimmler also describes a similar method for working upstream by piping over the sides. Tailings are hydraulicked away from the discharge of the sluice in both cases. This method is applicable to small, shallow deposits of low sluice grade.

Disposal of tailings by hydraulic giant is advantageous where sluice grades are low. Wimmler gives examples where tailings containing up to 15-in. rocks have been stacked 52 ft. high by a 4-in. giant with 310-ft. head; in another instance, stones up to 12-in. size were stacked 35 ft. high by a 3-in. giant with 100-ft. head. He states that the distance over which the pile can be stacked approximates one-half and the height one-third of the head at the nozzle. The Ruble elevator is another device to facilitate stacking. It consists of a 20-deg. chute, 8.5 ft. wide by 52 ft. long, the sides of which are 10 ft. at the lower end and 6 ft. at the upper end. The lower 22 ft. have a solid bottom; the upper 30 ft. have a bottom of steel-shod grizzly bars, 2.5 by 5.5 in., set transversely. A riffled table receives the underflow of the grizzly and is designed like an undercurrent. The elevator is mounted upon skids or rollers so as to be movable about the floor of the pit. Two giants are used, one to drive the gravel over the bedrock to the elevator and the other to drive it up the slope.

The quantity of water required in hydraulicking varies between wide limits. With favorable bedrock and sluice grades of 4 to 4.5 per cent, maximum duty is obtained and ranges from 3.5 to 7 cu. yd. per inch per 24 hr. Under more adverse conditions duties of from 1 to 2.5 cu. yd. per miner's inch are not uncommon. Ordinarily water is supplied under natural head, but there are a very few instances where power pumps have been successfully used.

J. P. Hutchins gives notes on the use of pumped water on the Nechi River by the Pato Consolidated Gold Dredging Co. (South America) for working relatively small areas of gravel from a few feet to 30 ft. and averaging 15.5 ft. in thickness. There are no large boulders; decomposed bedrock is 15 to 125 ft. above the river. The jungle is cleared by bulldozers at a cost of $400 per acre. Sluices are made in 8-ft. sections so as to be readily portable; they are constructed of 8-gage plate, bent to shape. Riffles are 3- by 3- by ⅝-in. steel angle spaced tranversely 2¼ in. apart, canted against stream flow and welded in sections. The sluice length is 20 to 500 ft., with grade 3½ to 4 in. per 8-ft. section. Water is pumped by 12-in. centrifugals direct-connected to motors and carried upon barges in the river; pump rating is 4,000 gal. per min., 200-ft. head. Connections are made with the 16-in. shore lines by sections of high-pressure rubber hose. The 16-in. pipe lines are in 30-ft. sections of spiral-lap-welded pipe with bolted couplings; secondary or branch lines are 10 and 12 in., and giants are equipped with 3-, 3½-, and 4-in. nozzles. The average life of each working is 1 year. From 1938 to 1941, 3,200,000 cu. yd. was pump-hydraulicked; annual yield, respectively, was 25.6, 22.7, 35.6, and 47.3 cents per cubic yard, with recovery estimated at 85 per cent. Power cost of pumping was 3.86 cents and direct expense for hydraulicking was 15.17 cents; total pumping cost was 4.48 cents per cubic yard; power cost at pumps 0.4 cents kilowatt-hour; average cost, including government royalty, was 21 cents per cubic yard.

Hydraulic mining costs vary widely and are greatly influenced by local conditions. At one extreme may be placed the California costs of the old days, where under especially favorable conditions gravel yielding 2 to 3 cents per cubic yard is said to have been profitably worked. Bowie gives costs for 1876 of the La Grange Company as 6 cents per cubic yard, yield 10.19 cents per cubic yard; water duty 1.48 cu. yd. per miner's inch per 24 hr.; average value of gold and silver $19.29 per ounce. At some North Bloomfield operations, costs ranged from 2.86 to 6.19 cents per cubic yard and yields from 13.7 to 23.8 cents per cubic yard; water duty ranged from 3.86 to 4.8 cu. yd. per miner's inch per day; sluice grades were 5.5 per cent. Examples of costs given by Wimmler for Alaskan conditions are as follows: Average depth of deposit 9 ft.; hydraulic equipment and 1,600-ft. ditch cost about $5,000; average costs 30 to 35 cents per cubic yard. At another operation equipment costs were about $30,000 and mining cost 43 cents per cubic yard; at a third mine, costs ranged from 33 to 65 cents per cubic yard.

Opposition to hydraulic mining debris and stream pollution closed down hydraulic mining in California in 1884. Comparatively recently, debris-restraining dams have made possible some hydraulic mining; but since 1933 the development of this kind of mining has been slow, despite the increased price of gold, because of fear of legal entanglement.

However, hydraulic methods of mining are used in many placer regions for mining gold, tin, platinum, and other minerals. In phosphate mining, hydraulic methods have been widely used for stripping under favorable conditions. Dredge pumps and pipe have also been used for transportation of the material. As an excavating and transportation device, pressure-water methods yield the greatest output per unit of man-power, and costs are very low where water is cheap. The legal and other limitations placed upon its use are very severe and in most instances prohibitive.

Dredging.—Dredging and other similar backfilling methods are applicable to alluvial deposits that are of the basin, lake-bed, delta, terrace, and river-bed types. Dredging of alluvials had its inception in New Zealand about 1882; it was introduced into the U.S. at Bannock, Mont., in 1897. W. P. Hammon and Thomas Couch placed in operation the first successful gold dredge in California at Oroville in 1898, since which time the dredge has found application in many placer regions all over the world. Larger and deeper digging dredges have been designed until at present the deepest dredging operations are 124 to 135 ft. below pond level; with a bank above pond level, a deposit some 150 to 200 ft. thick can be mined by this method.

Two types of dredges have been used in mining, the continuous- or connected-bucket dredge and the suction dredge. The latter has never been successfully used in gold mining and is now probably no longer used in tin mining, where it found some application. Several different designs of the continuous-bucket dredge are in use. The stacker design is predominant, the flume design being a feature of smaller dredges used in Alaska and a few other alluvial districts. In the simplest type, the buckets discharge into a flume or sluice that projects over the stern of the barge and discharges its tailings into the pond; another modification employs a revolving screen to receive the bucket discharge. The oversize is dropped into a chute at the stern and the undersize is discharged into sluices on either side of the screen; tailings being discharged by tail sluices beyond the end of the barge. A third type discharges its screen oversize upon a belt conveyor that extends out beyond the end of the barge and builds up a low pile. The flume is a metal sluice with longitudinal or transverse steel riffles. Dredges of this type are of 1.5 to 3 cu. ft. bucket capacity and are used for shallow digging, 4 to 20 ft. below pond level. Many such dredges have been used in Alaska.

Dipper and clamshell dredges have been tried out in gold dredging, but all except the Becker-Hopkins have been unsuccessful. Steam, gasoline, and diesel power have been used for the operation of dredges, but electrical power is now almost universally used. In smaller dredges, diesel-engine units may be used for power. In other cases a power plant is constructed at a convenient location on shore.

The dredge is constructed in a prepared excavation, which is flooded before operations are started. Some dredge hulls are launched into their ponds and the machinery installation is completed upon the floating hull.

The Gold Dredge.—The gold dredge is a self-contained electrically operated digging, washing, and disposal unit, operating in a pond and digging below water as well as on a bank of moderate height above water. It backfills its pit as it advances the cut. Its machinery is mounted on a shallow-draft hull. The digging element is a chain of maganese-steel buckets carried by a ladder (plate girder or truss) with a cylindrical flanged lower tumbler and a shaft close to the upper hexagonal-shaped tumbler, which is supported by shaft and bearings carried by the framing. Figure 219 shows a plan and elevation of a large dredge.

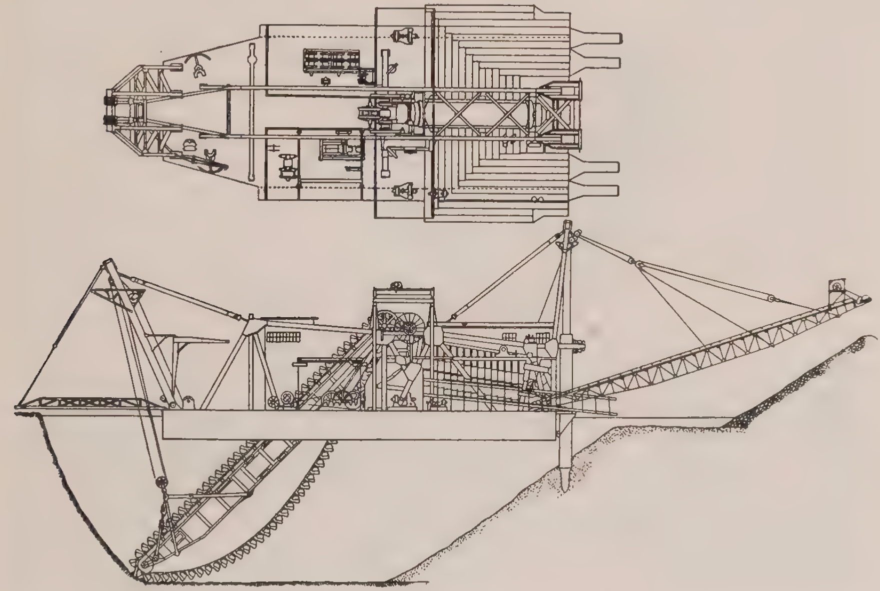


FIG. 219.—Plan and elevation of a 17-cu. ft. bucket gold dredge. (*Yuba Mfg. Co.*)

On the upper side of the ladder, the chain of buckets is supported by rollers at intervals and hangs beneath the ladder as a catenary. The ladder at its lowest digging position is usually at an angle of 45 deg. and when raised to its highest position is at a small angle below the horizontal. The upper hexagonal-shaped tumbler and its shaft are driven by one or two bull gears and a train of gears, motor or motors, and V-belt drives at a speed of 4 r.p.m., giving a bucket speed of 20 to 35 per min. A braking wheel and band brake are placed on one of the shafts of the gear train to hold the chain of buckets when power is off and when buckets are changed.

The buckets discharge into a hopper and chute and thence to a revolving screen with perforations ¼ to ⅝ in., which washes boulders, disintegrates clay, and delivers a more or less dewatered oversize to the stacker conveyor belt (Fig. 220) upon which it is carried up at an angle of 18 to 20 deg. to a discharge point above the tailings pile at the rear of the dredge, behind the pond. Water under pressure (70-ft. head) is sprayed by nozzles within the screen to break up the material discharged by the buckets. Undersize with its accompanying water is distributed over wide tables (28 to 32 in. in width, grade 12.5 per cent or more) equipped with Hungarian riffles, the tables being usually transverse to the screen axis. These tables discharge into side sluices that extend some distance beyond the stern of the hull and deposit fine material into the pond. Side spill of the buckets at the hopper falls upon the inclined grizzly. The undersize passes to the "save-all" tables (Fig. 221), and the oversize drops into the pond through the well.

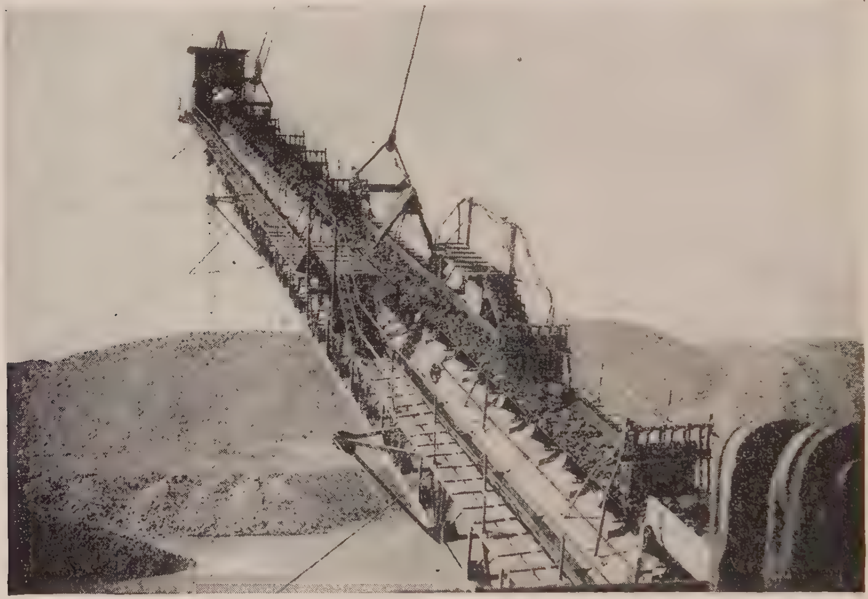


Fig. 220.—Stacker and belt conveyor of gold dredge.

The hull construction includes a well or narrow open space in which the ladder is raised and lowered. The well extends from a position forward of the bucket discharge to the bow of the boat. Spanning the well at the bow is the ladder gantry, which supports the rigging and blocks by which the ladder is raised and lowered. At the stern of the boat is the stacker gantry, which supports the stacker truss by blocks and wire ropes and also supports the rigging by which the heavy steel spuds are

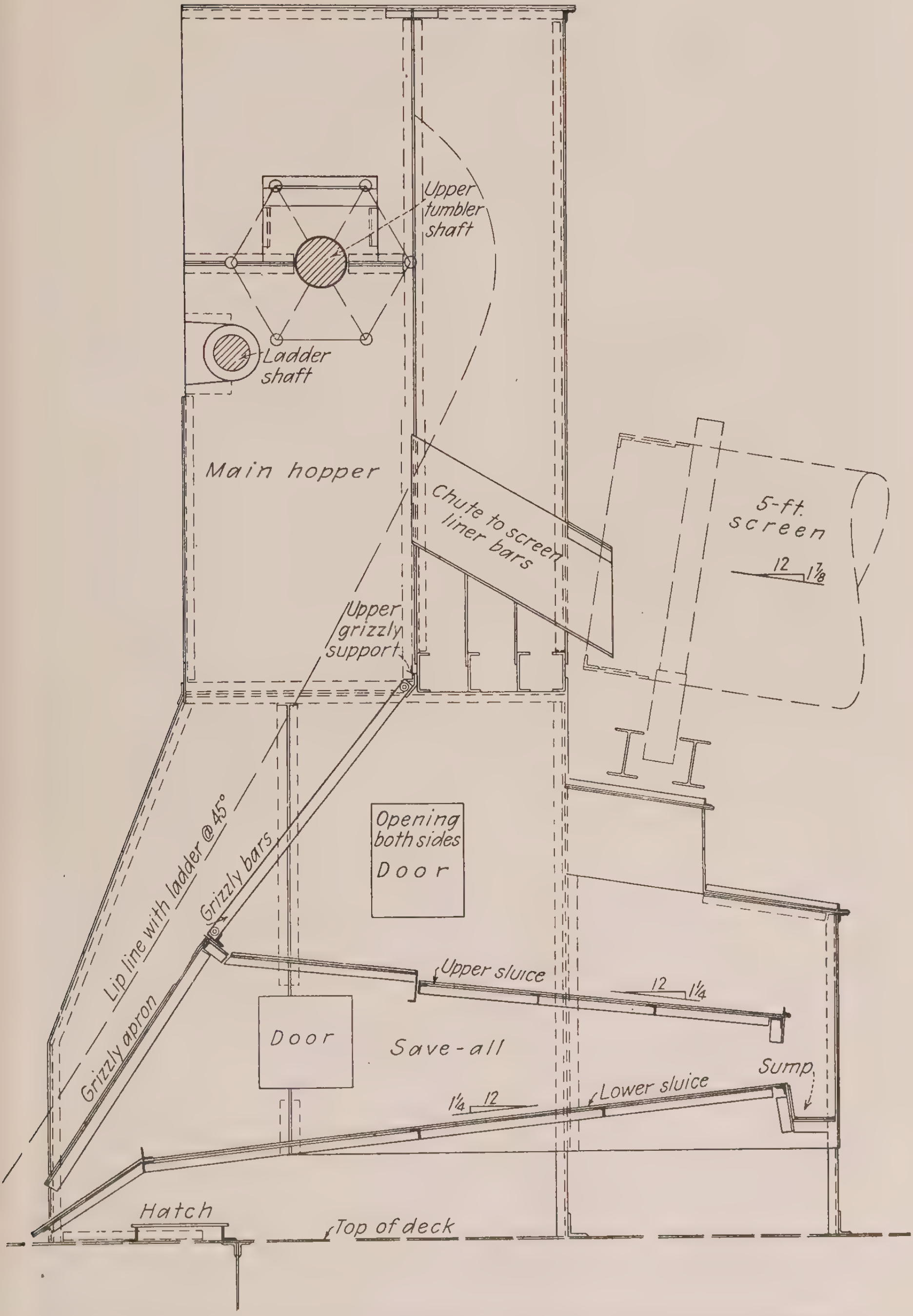


FIG. 221.—Save-all tables and grizzly at discharge hopper of gold dredge. (*Yuba Mfg. Co.*)

raised and lowered. Longitudinal steel trusses support the machinery, stiffen the hull, and afford ties for both gantrys.

Accessory equipment on the hull includes the ladder hoist; multiple winch (five to eight drums for swing lines, spud, stacker, etc.); high-pressure, low-pressure, and cleanup centrifugal pumps, each direct-driven by motor; and transformers and control apparatus. A traveling crane is placed over the discharge end of the bucket line, and jib cranes are placed at the bow gantry for handling dredge buckets. Power is brought to the boat by a suspended armored cable, or a cable is supported on pontoons from a shore connection. A power-generating unit, usually diesel electric, is sometimes incorporated in the dredge design.

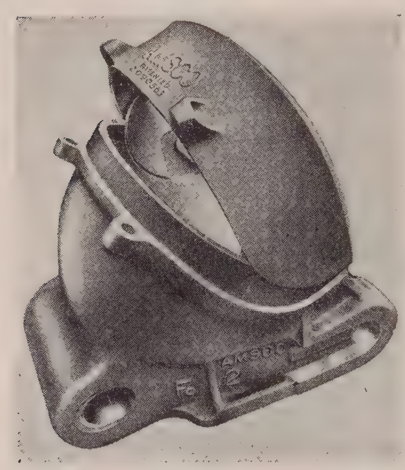


FIG. 221*a*.—Rivetless lip gold-dredge bucket; 10.5 cu. ft., weight 2,735 lb., bushing 56 lb., pin 330 lb., total 3,121 lb. (*Wisler, American Manganese Steel Co.*)

Lines at the bow are operated by the winch for swinging the boat about the spud as a center when digging. Such lines are attached to the lower end of the ladder on either side and extend to shore sheaves, which are anchored to deadmen and return through sheaves to the winch drums. In some rigs the bow lines pass from the drums to sheaves on either side of the ladder end and thence to anchorages on shore. All controls of winch drums, ladder hoist, brakes, and other machinery are concentrated in the operating room of the dredge, which overlooks the bow of the boat, the bucket line, and the pond at the bank cut. Two stern lines are used for shifting the dredge, but these are slacked off when the dredge is making a cut. The dredge can also be walked forward by using the spuds alternately if two are provided. In other cases a separate head line is used. In river dredging the head line is generally used.

Gold dredges are rated according to bucket size, and their proportions, power, and other design features depend upon this as well as depth of digging, kind of gravel, digging conditions in general, and climatic conditions in which they are to be operated. Dredge design is standardized as far as possible, but the important features of the dredge are in accordance with its particular work. Bucket sizes are 1¼ to 2½ cu. ft. on the smallest and 3, 5, 6, 7½, 9, 10, 13½, 15, and 18 cu. ft. on the larger dredges. Buckets are bowl-shaped and are made with replaceable lips held by two bolts (Fig. 221*a*). They are joined by massive bucket pins of alloy steel. Bucket eyes are bushed to allow wear replacement.

Depth of digging, which is a controlling feature, ranges from 4 to 124 ft. below pond level. Ladder length is dependent upon digging depth and maximum ladder angle. The deepest digging on record is by a 12-cu. ft. dredge described by F. G. Payne, which recovered tin from 120 to 135 ft. below water line. The ladder length of this dredge is 195 ft. Yuba 20, a California dredge, has a digging depth of 124 ft. at a ladder angle of 45 deg. and a ladder length of 216 ft. The cost of a dredge is closely related to bucket size and digging depth and whether wooden or steel hulls are used. A dredge of the Yuba 20 type (18 cu. ft.) represents a prewar cost of about $1,000,000.

TABLE 103.—EXAMPLES OF DREDGES

Bucket capacity, cu. ft.	3	5	10	4.5	18
Bucket speed, buckets per min	26	22	22	32	21
Hull dimensions, ft.	85 × 44 × 6	110 × 44 – W	167 × 74 × 12	90 × 41 × 7	250 × 80 × 11
Digging depth, ft.	20	40	60	20	124
Buckets in line	65	81	106	76	135
Stacker length, ft.	80	100	110	85	225
Spud length, ft.	....	....	....	44.6	70
Number of spuds	2	2	2	1 – 10t	2
Screen size, ft., dia. and lgth.	5 × 30½	6 × 36	9 × 59	5 × 32½	9 × 50½
Table area, sq. ft.	1,260	1,085	4,800 – J	....	6,000
High-pressure pump, in.	8	8	14	8	14
Low-pressure pump, in.	8	8	8	8	14
Capacity of pump, g.p.m.	2,000	2,000	6,000	2,500	5,500
Power load, hp.	177	285	300	335	2,175
Bucket drive, hp.	75	100	300	100	(2) 300
Average daily yardage	2,800	3,500	8,000	4,400	12,000

NOTE.—First three are Fairbanks Exploration Alaska dredges; the fourth is a French Gulch dredge (data taken from *Mining World*). The fifth is Yuba 20, description by H. Sawin in *Engineering and Mining Journal*. *W* is wooden hull; all others steel hull. The French Gulch dredge was constructed of pontoons. *J*—16 jigs were installed in addition to tables. In addition to high-pressure and low-pressure pumps, most dredges are equipped with a stand-by and other pumps. Yuba 20, in addition to the two main pumps, was equipped with a 6-in., 1,100-g.p.m., 112-ft. head pump; a 4-in., 450 g.p.m., 65-ft. head pump; and a 2-in. service pump. The cost of Yuba 20 was said to be $1,000,000 and its weight some 3,700 tons.

In the deeper digging dredges an idler pulley 9 ft. in diameter is placed beneath the ladder just below the bucket discharge so as to give bucket clearance at the hopper and to prevent slack buckets from dragging upon the bottom. On very deep-digging dredges the catenary is supported approximately at mid-point to avoid excessive bottom drag and to distribute weight. The Perry idler, invented by O. B. Perry for this purpose, is some 10 ft. in diameter and weighs 40 tons with its attachments.

Yuba 17, with 112 ft. depth of digging, has a bucket line of 126 buckets, each weighing 2½ tons, and a total weight of 315 tons. A Perry idler is used on this dredge and also on other deep-digging California dredges.

Three types of hull are in use. The wooden hull is most suitable for use in arctic and subarctic areas and the steel hull for tropical conditions. Most dredges are now constructed with steel hulls. The pontoon type of steel hull is widely used, especially where the deposit to be dredged is small and it is planned to dismantle and reerect the dredge upon another area after exhaustion of the first. The first pontoon-type hull, constructed by Yuba in 1933, was for a Montana property. The hull consisted of 33 pontoons, electrically welded and then bolted together and provided with reinforcement at points where machinery and frames are supported. Pontoon construction reduces both labor and time in erection and in dismantling. Approximate hull sizes are given in Table 103.

A-c power is almost always installed. There are some d-c installations with Ward-Leonard control. Power requirements vary with the size of the dredge and the depth of digging. A 3-cu. ft. dredge has an installed horsepower of 159; a 9-cu. ft., 567; an 18-cu. ft., digging 80 ft., 1,850; the Yuba 20, 18 cu. ft. and digging 124 ft., 2,200. Digging power requirements range from a low of 0.62 kw.-hr. for shallow easy digging with low bank up to 2.4 kw.-hr. per cu. yd. for more adverse conditions. For ordinary conditions the range is from 1 to 1.5 kw.-hr. per cu. yd.

Dredge capacity depends upon the kind of material dug. Loose free gravel gives maximum capacity, and compact cemented gravel, imbedded in stiff clays with large boulders in excess, gives lowest rates. Bucket speeds, which ordinarily range from 20 to 30 buckets per minute, is another factor. In some recently constructed dredges, with variable-speed motors or d-c motor drive, bucket speeds up to 38 per minute have been or can be used. Bucket speed and dredge capacity are also tied in with recovery. Maximum yardage that can be efficiently treated upon tables or jigs with minimum loss in tailings is the prime objective, not maximum digging. Operating time ranges from 75 per cent or less under difficult digging up to 93 per cent under favorable conditions. Dredge capacity ranges from 600 to 800 cu. yd., place measure, per cubic foot of bucket capacity, with operating time 80 to 85 per cent of total time. Very deep digging gives lower capacity; an 18-cu. ft. dredge has a normal capacity of 12,000 to 15,000 cu. yd. per 24 hr., but when digging at 124 ft. depth, the capacity is reduced to 10,000 cu. yd., largely because of the inability of the operator to determine the digging feel of the buckets at such depths. Likewise, shallow digging gives lower capacity, as the frequency of stepping ahead is greater. Undermining and by-passing valueless gravel in the bank by laddering over gives greater area covered than normal, although some of the by-passed gravel on the bedrock is necessarily dug and passed through the dredge.

Dredging costs vary with the size of the dredge, depth of digging, compactness of gravel, proportion of boulders, efficiency of the dredge crew, power cost, repair and replacement costs, and climatic conditions. For smaller dredges in California costs range from 6 to 8 cents per cubic yard; larger dredges operate at about 3 to 5 cents per cubic yard. For deep digging, lowest costs are at 100 ft. in depth and for deeper digging the cost gradually rises. Under Alaskan conditions with short seasonal operation, small dredges, and thin deposits, Wimmler gives operating costs from 15 to 35 cents per cubic yard, with 15 to 18 cents per cubic yard for well-managed properties. For Nome, where thawing may be necessary, the cost is estimated at 23 cents per cubic yard for a seven-months season.

A single-bucket dredge, the Becker-Hopkins, has recently been placed on the market. It is suitable for shallow deposits, 8 to 10 ft. in thickness. The dredge consists of a barge carrying a revolving screen, tables, stacker, and other accessories. Forward is an A frame and well. The bucket is scoop-shaped and carried at the end of a U-shaped telescopic trough, which is raised and lowered by blocks and wire rope supported by the A frame. The bucket is lowered into digging position, then raised, its contents sliding through the U-shaped trough to the screen at its rear. It is held against the face by cables. The cut is deepened by successive thrusts until bedrock is reached, when the barge is swung into a new position and a series of cuts is made to bedrock. One- and two-cubic yard buckets represent the sizes now used. A 1-cu. yd. dredge of this type is said to have attained a maximum digging capacity of some 170 cu. yd. per hr. Only a few dredges of this type are in use. There is little information about its capabilities, but there seem to be possibilities for it on small placer deposits.

Dredge Operation.—The operation of the continuous-bucket dredge consists of stepping ahead 6 ft. or more with the ladder elevated, then swinging the dredge by the bow lines at 30 to 50 ft. per min. about the fixed spud as a center, cutting the top portion of the bank in an arc the width of the pond or of the cut. The ladder is lowered sufficiently to obtain full buckets, ½ to 1½ ft., and the dredge is then swung back over the arc. The swinging of the dredge from side to side with successive lowering of the ladder is continued until the lower limit (bedrock) is reached (see Fig. 222). The bedrock is cleaned up, the ladder is elevated, and the dredge is stepped forward to begin a new cut. A line of full buckets indicates digging at full capacity. The winchman who controls digging adjusts his ladder, lowering it to keep the buckets full while maintaining the full speed of swing; he judges digging conditions by the opera-

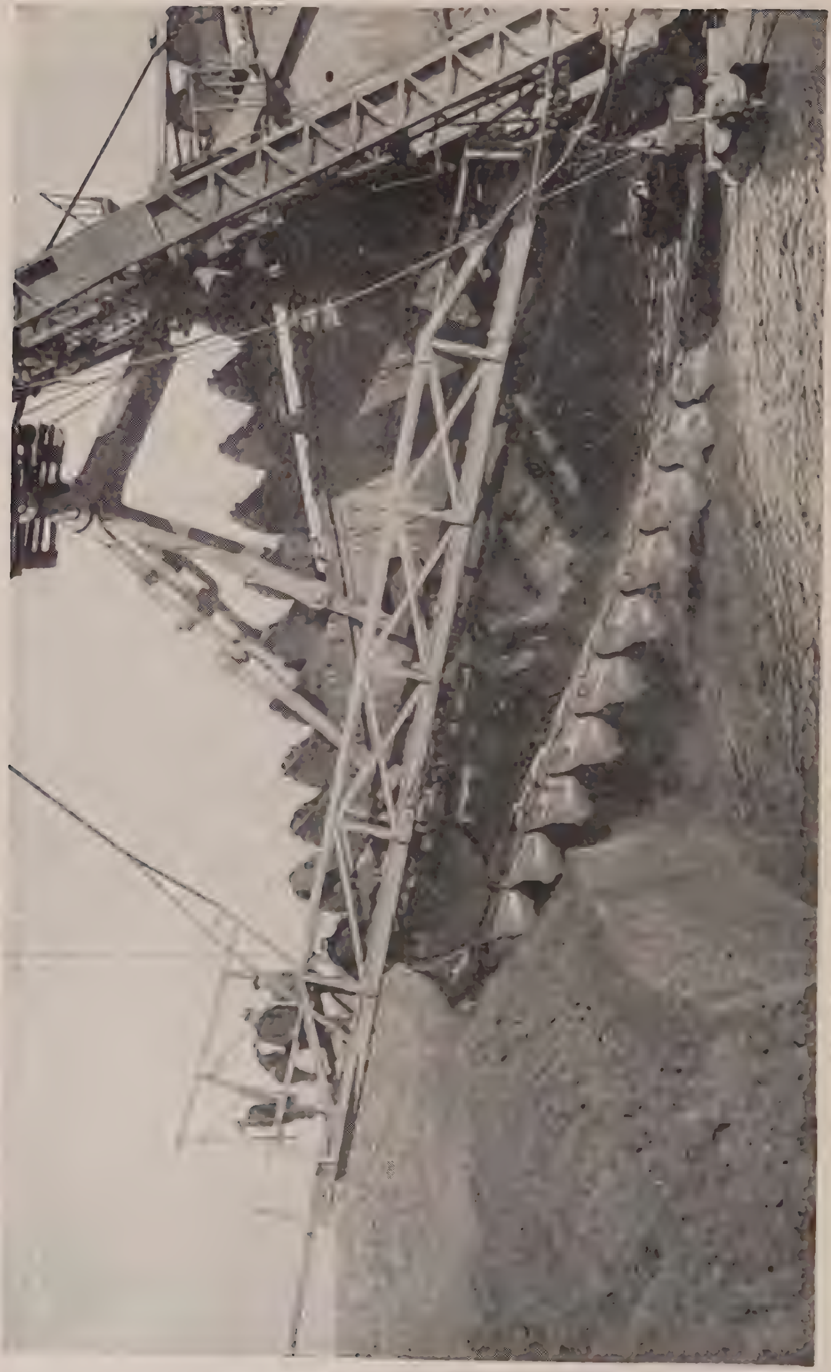


FIG. 222.—Dredge cut at its start; ladder is elevated and dredge is being swung.

tion of the bucket line. When operating against a bank above pond level the dredge undermines the bank and caves it into the pond, or a hydraulic giant is sometimes used to cave and gradually work down the bank. Banks up to 50 ft. above pond level have been handled in this way. In river-bed placers a head line is necessary where water currents are swift.

Often a parallel cut is made alongside the first (Fig. 223) and thus a pond of twice or more the width of the first cut is carried in wide deposits. The width of the deposit sometimes controls pond width; in other cases a pond 500 ft. or more is carried; in very deep digging, a wide pond is necesary on account of the length of the dredge and the necessity for maneuvering space. On wide deposits the dredge operates across the width of the deposit, turns at right angles, and cuts along the length until sufficient space has been excavated for turning. Then the next cut, paralleling the first, is started alongside the first.

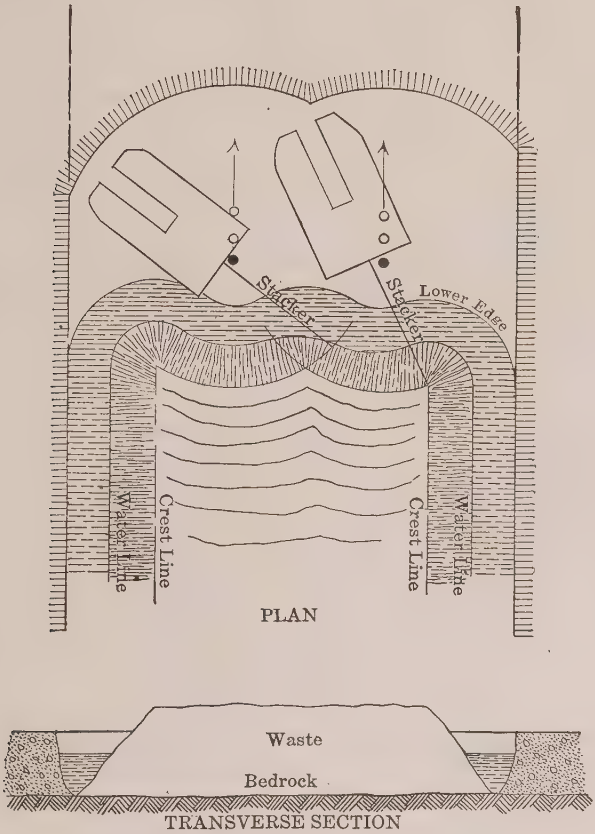


Fig. 223.—Plan of dredge cut. (*After J. P. Hutchins.*)

In narrow deposits the dredge makes only one cut. This may be upstream or downstream depending upon local conditions, but preferably upstream. Under such circumstances dredge cuts follow the pay channel, the position of which is established by previous sampling. River operations are controlled by the conditions of the bedrock and the stream. In bench areas dredges have been worked upgrade by depositing material in dams below the dredge cut so as to maintain a high pond level.

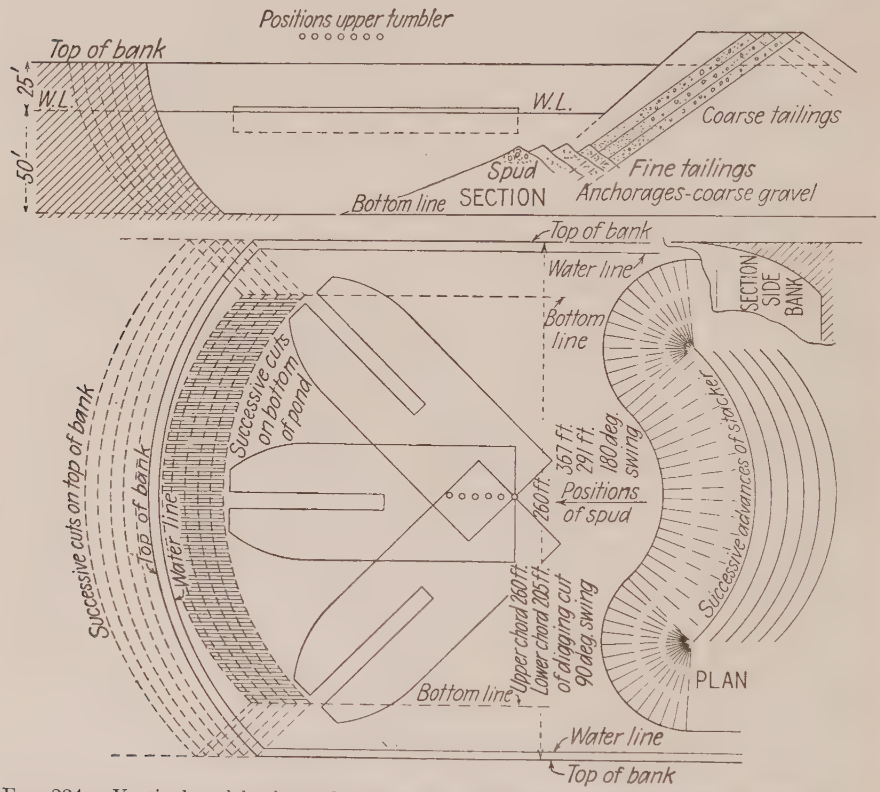


Fig. 224.—Vertical and horizontal cutting arcs of bucket line for successive positions of spud.

The angle of swing in finishing a cut is about 60 deg. to the right or left of the middle. The radius of cut is the length from spud center to bucket lip, and the cut advance may be from 6 to 15 ft. It is obvious that the radius is at the maximum when the ladder is raised to its uppermost position and is least when at its lowest reach with the ladder at a 45-deg. slope. A 100-ft. length of ladder gives 70 ft. at a 45-deg. slope (see Fig. 224). The bottom cleanup area is less than the top-cut area. Parallel cuts will thus leave a ridge in the pond bottom at the sides of the cut unless the angle of swing is increased at the bottom of the cut so as to dig corners and side ridges. The winchman can tell by the feel of the buckets or by the ammeter readings when he has cut through the ridge or corner at the side.

With high banks above pond level it may be difficult to make a swing sufficient to cut corners, and under such circumstances dredge cuts have to be overlapped sufficiently to take out intervening ridges. Soundings are taken for pond depths at one or more points during each shift and are reported by the winchman. At each cleanup or at least once a month the surface position of the cut is surveyed from the coordinate points that are established at 100- to 200-ft. intervals in both directions; flags give line points for the guidance of the winchman. Yardage is computed from average depths obtained from soundings and area advanced in a given time.

Deadmen or shore anchorages are changed at intervals. They may be placed as close together as 100 ft., or under favorable conditions may be 1,000 ft. apart with a bridle rig between. The bridle rig enables the bow-line attachments to be shifted as needed without change in the anchorages. In working wide areas, deadmen are placed well away from the pond and a long lead line is used. The size of the dredge, the material being dug, and the depth influence the width of the cut; a 6-ft. dredge in 25- to 30-ft. digging depth will give a minimum width of 140 ft. and usual widths of 200 to 210 ft.; a 17½-ft. dredge, minimum width 200 to 210 ft. and usual widths 250 to 350 ft., according to depth and tailing disposal (G. S. Dyer). Cleaning up the bottom depends upon depth and number of buckets dragging upon the bottom; from 4 to 8 buckets drag at 25 to 35 ft. depth, and 8 to 10 at 90 ft. Stepping-up distance is dependent upon ground, depth, tailing disposal, and bottom cleanup. Anchorage footing for holding the spud may sometimes be insufficient, and under such circumstances the dredge operator diverts the screen discharge through the chutes at the stern until sufficient spud anchorage has been built up.

Tailings are deposited on the pond bottom beyond the stern of the dredge and the screen oversize is stacked beyond the pond at a height limited by the stacker length and slope. Shoaling of the pond beneath the dredge is sometimes produced by excessive amounts of fines, or it sometimes results from the superimposed weight of coarse tailing crowding the fine sand tailing forward, especially where much clay is present in the fines. Two methods are in use to prevent this trouble: In one, the table tailings are delivered into sumps from which bucket elevators elevate the partially dewatered sands high enough to be delivered to the stacker conveyor; in the other system, sand wheels dewater and discharge the sand upon the stacker belt.

A water supply of 1,000 gal. per min. or more is required to maintain pond level in dredging. Total pond inflow is also dependent upon the yardage dredged per day, the porosity of formations enclosing the pond, and sometimes upon the necessity of diluting the slimes that accumulate in the pond. The dredge draws its supply for washing from the pond and all this returns to the pond except the relatively small amount retained on the oversize material. Pond level is maintained by weirs and ditches, and pumping may be necessary.

Very deep digging has its own peculiar problems. Below depths of 60 ft. the bucket catenary requires additional support; and the use of the Perry idler to divide it into two parts has improved mechanical operations of the dredge. Longer hulls are used for deep-digging dredges, and this entails an intermediate conveyor belt between the screen discharge and the stacker belt. Slumping of pit walls also sometimes occurs, and this possibility requires close control of digging operations at all times. Where much fine material is encountered, the fines assume such a flat slope as to run back along the bedrock to the digging position. The redredging of this material cuts down capacity and lowers recovery relative to yardage to an embarrassing and sometimes a mystifying extent. Frequent soundings are necessary to discover this condition. Yuba's mud pump overcomes the difficulty. The suction take-off is behind the digging position and the suction pipe is attached to the ladder; the special centrifugal and its direct-connected motor are on the deck close to the well, connections being made with pressure hose; the discharge line is carried to the stacker end; and the effluent is thrown well beyond the top of the rock pile or sent to a distant sump.

Difficult digging, tight ground, cemented gravels, and a large proportion of boulders are overcome by changes in dredge design. For difficult digging, a heavier ladder, heavier spuds, and oversize drive are used with a smaller and heavier bucket. Heavy wearing plates are placed on the bottom of the ladder so as to permit it to ride upon the bucket line in tight ground. A somewhat longer bucket pitch and a flattened bowl-shaped bucket with lip design that prevents wedging of boulders between buckets are employed with large, angular boulders. Rock eliminators are used to by-pass large boulders at the discharge point; and where this is impracticable, heavier screen plates and closer spacing of stacker-belt rollers enable such stones to be passed through the screen and up the stacker to the tailings pile. Difficult digging reduces the life of buckets and lips and increases the general wear and tear on dredge equipment, thus increasing costs materially. In earlier operations tight ground with many boulders could not be handled but better dredge design and construction have overcome many of these difficulties. Irregular bedrock, pot holes, and pinnacles of bedrock are other sources of trouble.

The gold dredge has been highly developed as a machine, and modern designs more than fulfill the tasks imposed upon them. C. M. Romanowitz is of the opinion that in the dredge of the future electronic devices will be applied to give better control in overcoming variations in digging conditions, to utilize electric power more completely, and to operate safety controls when the bucket line strikes submerged objects such as boulders and logs. Such devices will be particularly applicable to deep-digging dredges.

Auxiliary excavation equipment has been found advantageous in some recent dredging operations. At Lancha Plana (California) C. Patmon made use of tractors for stripping and tailing disposal in connection with dredging operations. At Manhattan (Nevada), according to N. Cleveland, tractors, bulldozers, scrapers, and carry-alls were used to strip overburden and draglines were used to clear tailings away from the dredge pond. A higher channel, out of reach of the dredge on a bench alongside the dredge cut, was stripped of its overburden, and the bedrock gravel, 10 to 15 ft. in thickness, was excavated by scrapers and dumped in the path of the dredge.

Frozen Ground.—In arctic and subarctic regions gravel may require thawing before it can be dredged. At Nome, Fairbanks (Fig. 225), and Dawson cold-water thawing technique has been highly developed. Present methods involve dragline excavation where ground is not frozen, and hydraulicking and washing away frozen muck that often lies above gravel beds. Thawing requires a water supply, a pipe system for its distribution, pumps, and accessories. Points, double extra-strong ¾-in. pipe with steel chisel-shaped ends, perforated for water circulation, are driven into the gravel in 10-ft. lengths. They are connected to water-supply pipes by lengths of hose. The points are manipulated into the gravel, the circulating water thawing the ground sufficiently to enable short depths to be attained from time to time. Points are spaced at the apexes of equilateral triangles, usually 16 ft. on a side, though spacing at 12 ft. is sometimes required. Driving is done by hand with a 25-lb. hammer that encircles the pipe and strikes against a hinged anvil. The anvil is bolted to the pipe and is equipped with wooden handles for turning the pipe. The point driver takes care of 20 to 30 points, driving each pipe to the depth of the thaw at intervals as required.

A unit area with its pump, sump, screen, and pipe distribution comprises from 1,200 to 1,500 points. At Fairbanks, point drivers average 10 ft. per man-hour and water flow 4 to 5 gal. per min. per point. The period of thawing ranges from 3 to 10 weeks; water duty approximates 10 cu. yd. per miner's inch per day. Gravels more than 50 ft. thick are drilled in winter by Keystone churn drillers using 3-in. bits. At Dawson, driving rates are 2 to 15 ft. per man-hour and water duty 5 to 15 cu. yd. per miner's inch per day. The capital expenditure for pumps,

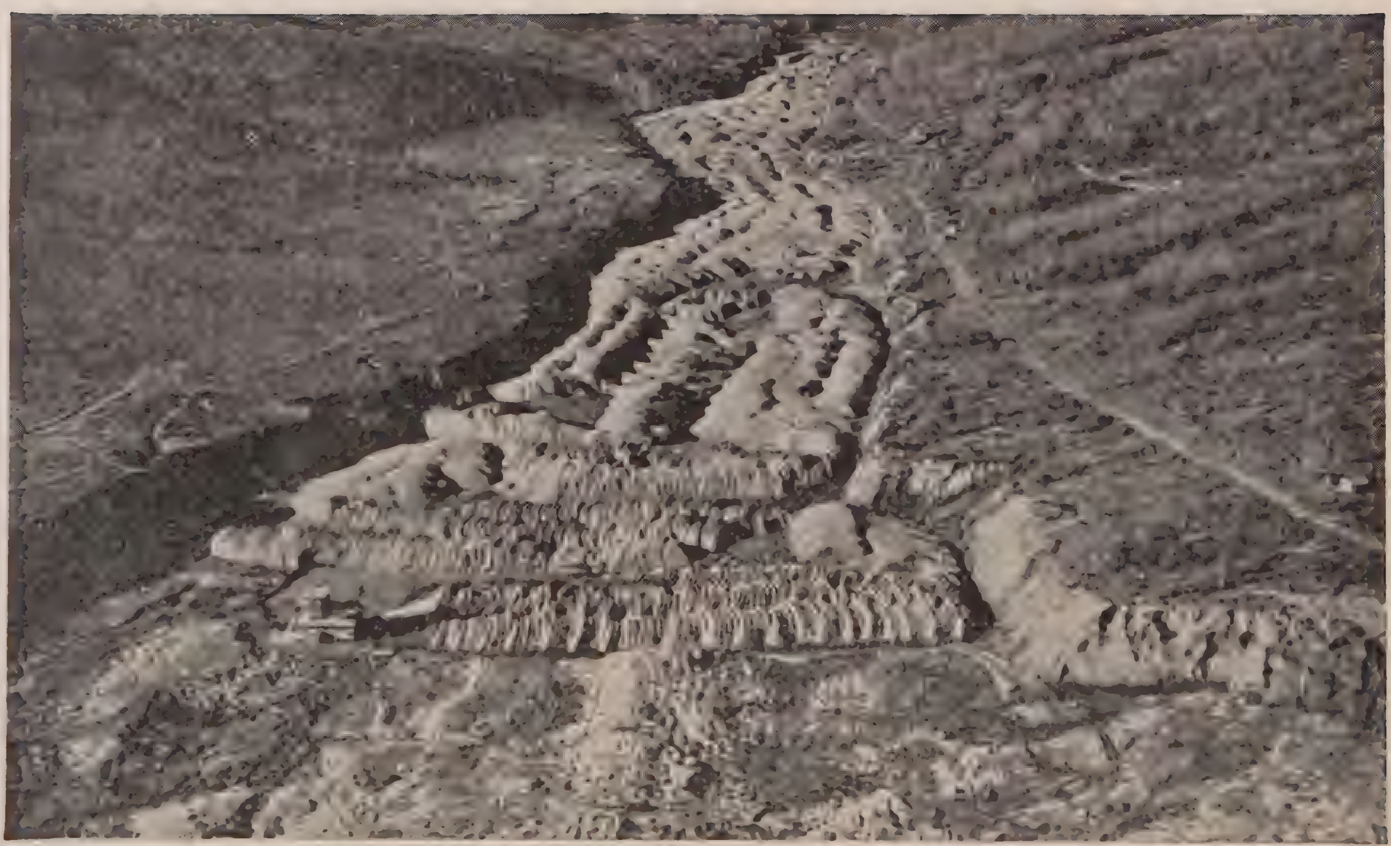


Fig. 225.—Cripple Creek dredge, Fairbanks Exploration Company—showing placer area, successive cuts, and dredge pond, dredge at left
(*Pacific Aerial Surveys, Inc.*)

power plant, pipe, hose, and water supply is very large. At Dawson, McFarland states that the capital investment for nine dredges, stripping and thawing equipment, power plants, buildings, and ditches represents $6,500,000. In 1938, overburden stripped was 2,818,000 cu. yd.; ground thawed, 4,141,000 cu. yd.; and ground dredged, 8,551,000 cu. yd.; power used was 28,073,000 kw.-hr.

The recovery of valuable minerals in placer operations follows much the same pattern everywhere. After digging comes disintegration, as thorough as possible, and then the separation of coarse oversize by a revolving screen, grizzly, flat screen, or other method. Clays are difficult to disintegrate, as they must be broken up and dispersed in water for thorough separation of the heavy particles. Beaters, log washers, or special tumbling sections on revolving screens are sometimes necessary. Clay lumps often go through to the tailings pile, and hence examination of the material there furnishes the clue to the efficiency of disintegration. The presence of nuggets requires tail-screen sections with larger perforations than the conventional ⅜- to ⅝-in. size, and a larger undersize is delivered to the tables at these points.

Treatment of fines is by save-all tables and Hungarian riffled tables, by diaphragm-type jigs, rarely by Ainley bowl, or by both jigs and tables. Quicksilver is used to improve retention of fine gold by amalgamation. Flour gold is difficult to save. Overloading of tables or jigs by rapid digging speed is avoided.

The proportion of undersize ranges from 30 to 80 per cent of the total gravel. Riffles and jigs pass on the greater part of the undersize, but some is retained along with the heavy minerals, gold, amalgam, and quicksilver. The upper riffles are removed first in the cleanup and as much of the sand as possible is washed down over the lower riffles; then the heavy product is gathered up and taken to the cleanup plant, where gold and amalgam are separated from the black sands by pan, rocker, concentrating tables, or other methods. Tin concentrates may be further cleaned on concentrating tables. Platinum and black sand that carries value may be by-products of gold dredging, and in rare instances tungsten concentrates and gem or industrial stones are obtained.

Dragline Dredging.—Dragline dredging involves two separate units, the first the digging unit or dragline excavator and the second a barge upon which is carried a receiving hopper and its grizzly, a revolving screen, riffled tables, stacker and conveyor for screen oversize, pumps for water supply, and a diesel or electric motor for power. The excavator operates upon the bank above the pond in which the barge floats. This pond is smaller and much shallower than that for the continuous-bucket dredge (see Fig. 226). Bow and stern lines and small hand winches control the position of the barge and enable it to be moved into receiving position. The excavator digs with its bucket under water, elevates the loaded bucket, and swings it over the hopper, where it is dumped; about two buckets per minute is the usual speed. Jets of water wash the material upon the hopper grizzly and carry undersize to the screen, which delivers its undersize to the riffled tables and its oversize to the belt conveyor of the stacker. Both units are mobile; operating man power is on a minimum basis, three men per shift, although other labor units are required for repair, stripping, and other work. The capacity of the digging unit is high but relatively less than that of the dredge. A tread tractor with bulldozing blade is a necessary auxiliary for stripping, brush clearing, leveling, and other similar work.

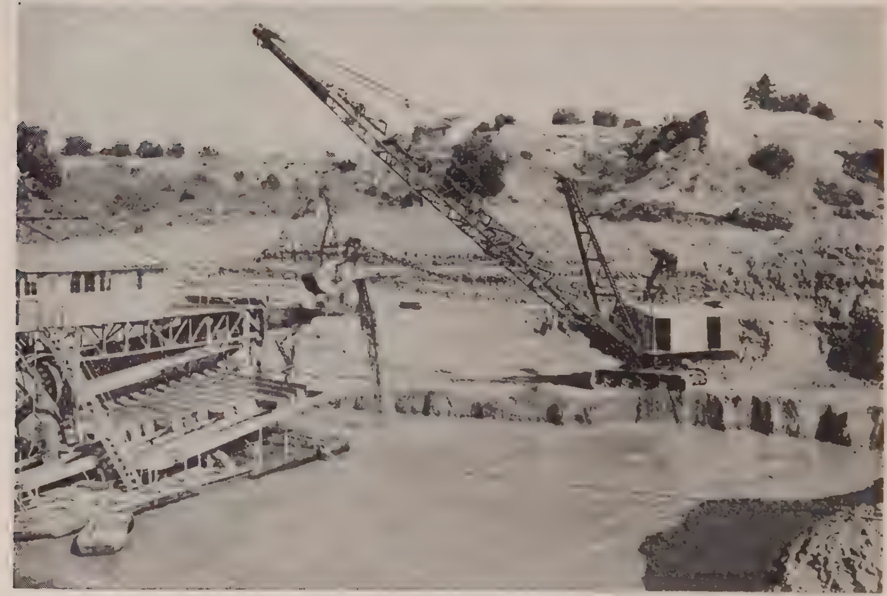


FIG. 226.—Dragline dredge and pond. (*Bucyrus-Erie Co.*)

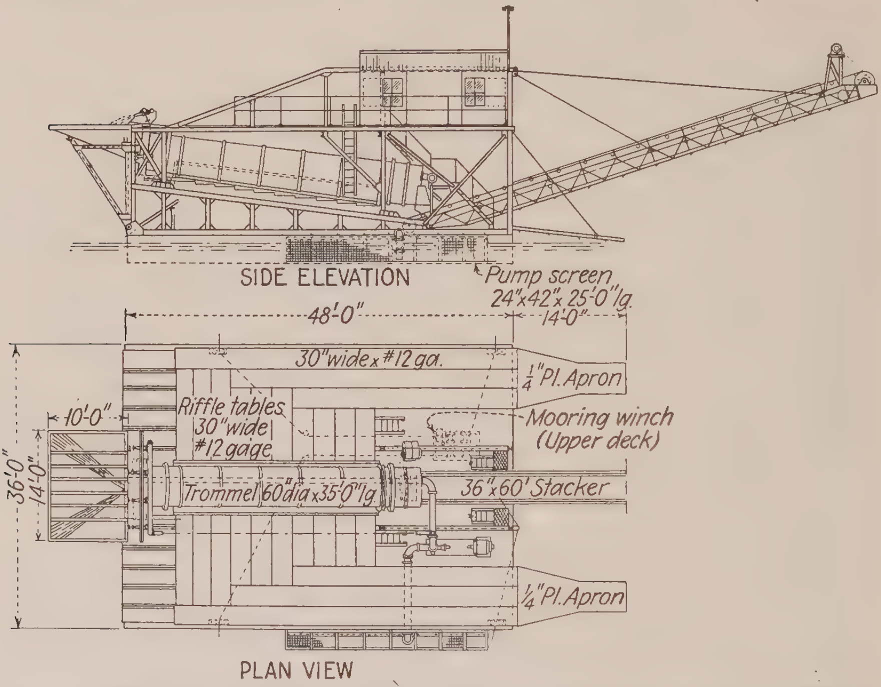
The dragline in combination with the floating washer was devised by J. P. Hutchins and first used in 1911 near the mouth of the Amur, eastern Siberia. In 1932, the Wyandotte Gold Dredge Company built its first dragline dredge in California, since which time some 200 dragline installations have been placed in operation in numerous placer districts. Next to the continuous-bucket dredge, the dragline dredge is the most widely used method for working small placers where low sluice grades or other reasons prohibit hydraulicking.

The dragline excavator is tread-mounted and provided with diesel or electric motor drive. Size range is from ¾- to 1½- cu. yd. bucket size and boom lengths 45 to 60 ft. The majority of operations make use of an excavator with 2- to 3-cu. yd. buckets and 60- to 75-ft. booms; the largest are equipped with buckets 5, 7, and 14 cu. yd. capacity, the booms ranging up to 185 ft. for the 14-cu. yd. bucket unit. Two types of buckets are favored, the Page and the Esco. Both are equipped with replaceable digging teeth. Most draglines are equipped with buckets of two sizes, one for average digging and the other and smaller one for hard digging. The excavator swings a full circle, 4 r.p.m. The bucket can be lifted vertically and swung beyond the vertical when dropped; it can be drawn along the bottom of its cut and in a curving slope upward to the level upon which the machine rests; the boom slope can be varied from 15 to 70 deg. A 1-cu. yd. dragline, boom length 50 ft., has a dumping radius of 30 to 35 ft., a dumping height of 32 to 35 ft., and a cut below grade of 15 to 18 ft. A 2-cu. yd. 70-ft. boom has respective dimensions of 55, 43.5, and 22.6 ft.; a 2.5-cu. yd. 90-ft. boom, 80, 43.75, and 54.75 ft. Crawler travel speed is 1⅛ m.p.h.

Power-shovel manufacturers also supply dragline excavators, as both machines have many features in common. Width and depth of cut depend upon boom length. Most gravel deposits mined by dragline dredging are from 10 to 15 ft. in thickness, but excavators are made to dig to much greater depths. The width of cut is slightly more than double the dumping radius. The engine on a 1¼-cu. yd. bucket excavator with 60-ft. boom is a 102-hp. diesel. The normal bucket speed is two buckets per min. or 150 cu. yd. per hr. Buckets take about three-fourths of the rated capacity.

Barges are constructed of steel or wood, preferably of steel. The pontoon type of steel hull is now common. Smaller barges are 30 by 40 ft. by 3 ft. depth and draft may be about 2 ft. The largest hull constructed for this purpose was installed near Dayton, Nev., and was approximately 50 by 106 by 7 ft. It was constructed in 13 steel pontoons and weighed 770 tons with equipment. The 12-yd. dragline, 185-ft. boom, excavated to a depth of 87 ft. below pond level from a bank 30 ft. above pond level.

Screen, table, and stacker design follow closely the standards established by the continuous-bucket dredge. Figure 227 is a plan of the washer. In some installations jigs have taken the place of tables, and Ainley bowls are sometimes installed as gold savers. One of the first Wyandotte dredges, described by J. F. Magee, had a wooden hull 40 in. deep and draft 25 in. when fully loaded and displaced 65 tons. The equipment of this barge included the following: A steel receiving hopper of 10 by 10.5 ft. with a rail grizzly, bar spacing 14 in., hopper-bottom slope 25 deg. to screen chute and 30 in. wide; the screen, 48 in. in diameter by 25 ft. in length, welded construction, holes ⅜ in. on 1⅛-in. centers and operated at 14 r.p.m; a table area of 537 sq. ft. of which 480 sq. ft. was effective riffle area arranged in seven cross sluices of 27-in. inside width on each side of the screen, which discharged on a grade of 1½ in. per ft. into side flumes projecting 12 ft. beyond the end of the barge; a stacker with 24-in. belt, 45 ft. in length; a centrifugal pump with 7-in. discharge and a capacity of 1,200 gal. per min., of which 85 per cent went to the screen, 14 per cent to two nozzles in the hopper, and 1 per cent to a skimmer jet that cleared surface trash away from the pump-suction inlet screen; a diesel power unit, 50 hp., driving by leather belt countershaft, and screen, pump, and stacker by belt or chain drive; a 1,500-watt lighting unit; and a 3-in. centrifugal cleanup pump driven by gasoline engine.


SIDE ELEVATION
Pump screen 24"x42"x 25'-0" lg.
48'-0"
14'-0"
30" wide x #12 ga.
¼" Pl. Apron
Riffle tables 30" wide #12 gage
Mooring winch (Upper deck)
10'-0"
36'-0"
14'-0"
Trommel 60"dia x 35'-0" lg
36"x60' Stacker
¼" Pl. Apron
PLAN VIEW

FIGS. 227-228.—Elevation of barge and stacker; plan of barge, dragline dredge. (*Mining and Ind. News.*)

Magee gives an estimate of the water-sand ratio when digging at maximum capacity of 2½ cu. yd. per min: Assuming 40 per cent sand tailings and 60 per cent rock tailings, the sand flow amounts to 1 cu. yd. per min. and the water flow of 6 cu. yd. per min. gives a 15 per cent sand content of the pulp. The total length of the barge from hopper to end of stacker is about 75 ft.; the dragline cut and the pond are each about 70 ft. in width.

For successful dragline dredging a heavy bucket is essential. C. Thurman states that a 1½-yd. bucket should weigh not less than 3¼ tons and a 4½-yd. bucket 9 tons for effective digging. Replaceable teeth are an essential feature of bucket design. He also states that deposits are best worked by this method in narrow strips, 12 to 20 ft. in width, across the width of a deposit, the strips being taken in sequence upstream. In starting a new cut at the side of the pay area, the cut is made in the new strip and the bucket is also worked over the bedrock of the previous cut in diagonal sweeps. The pond parallels the bank slice; on the return cut the bedrock is worked in diagonal sweeps at an angle to the sweeps of the previous cut, thus working the bedrock in two directions, each crossing the other. Thurman finds that much of the success of dragline dredging depends upon working down bedrock as deeply as possible, as the principal pay is on, in, or close to bedrock. He also advises operators to dump the bucket, no matter how little its load, on each sweep and never to return the bucket to complete a full load.

Portability of equipment and low capital cost are advantages of dragline dredging. Equipment of the Wyandotte dredge, according to Magee, cost as follows: Dragline, 1¼-yd. bucket, 60-ft. boom, $18,000; washing plant, $12,000; accessory equipment, $10,000; total, $40,000. This outfit dug 557,764 cu. yd., 9.5 ft. average depth, from Apr. 5, 1935, to Jan. 1, 1936, or an average of about 60,000 cu. yd. per month, at a total cost of $50,406.48, or 9.04 cents per cubic yard. C. V. Averill gives costs for two plants in California: A 1¼-cu. yd. dragline with diesel power, $50,000; and a 3-cu. yd., electric power, $71,000. He also gives direct operating costs, Table 104.

TABLE 104.—DRAGLINE-DREDGING COSTS
(C. V. Averill)

	Diesel	Diesel	Electric
Dragline bucket, cu. yd.	1.5	1.5	3
Expense per cu. yd.:			
Dragline	$0.021	$0.023	$0.012
Washing	.019	.020	.013
General	.025	.028	.014
Total operating	0.065	0.071	0.040
Depreciation	.015	.018	.009
Total cost	$0.080	$0.089	$0.049
Per cent total time operated	49.6	56.8	61.2
Cu. yd. per hr.	181	133	260
Cost of dragline	......	$22,000	$30,000
Cost of tractor	......	6,500	8,000
Cost of barge	......	20,000	28,000
Cost, miscellaneous	......	1,500	5,000

NOTE.—Operating crew 13 men, as follows: 3 on dragline, 1 oiler, 6 on barge, 1 on tractor, 1 mechanic, 1 extra.

Table 105 is another example of cost.

TABLE 105.—OPERATING COST, 3-YD. DRAGLINE
(H. S. Lord, American Mining Congress)

Detail	Amount	Cost, cu. yd.
Pay roll	$33,800	$0.0264
Repairs	10,700	.0086
Gas, oils, fuels	8,600	.0069
Compensation etc	2,800	.0023
Other expenses	3,800	.0030
Insurance and taxes	1,200	.0009
Cable	800	.0007
Welding material	700	.0006
Freight and express	400	.0003
Total field cost	$61,800	0.0497
Depreciation	25,000	0.0201
General office	5,000	0.0040
	$91,800	$0.0737

NOTE.—Direct costs for 3-cu. yd. installation handling 1,243,000 cu. yd. in 333 working days in 1 year; no interest, income taxes, prospecting, moving expense, or royalties are included.

Operating or digging time in proportion to total time is far lower than for the continuous-bucket dredge, ranging from 50 to 65 per cent of the total time. Objections have been made to dragline dredging on the score of incomplete bedrock cleaning, and undoubtedly this is a fundamental weakness of many operations. There is often difficulty in washing where clays are encountered but this is overcome by changing the screen design and incorporating baffles and lifters in the upper nonperforated section of the screen so as to increase its disintegrating effect upon the material delivered by the hopper. The dragline is highly important and successful in working small, shallow deposits or steep gulches where scarce water and low bedrock grades preclude other methods. Its portability is of great advantage.

TABLE 106.—APPROXIMATE COST OF DRAGLINE DREDGE EQUIPMENT
(H. S. Lord, American Mining Congress)

Dragline bucket, cu. yd.	1½	2	3	5	12
Boat capacity, cu. yd. per day	1200–1500	2000–2500	4000–5000	5000–6000	10,000–12,000
Dragline	$25,000	$36,000	$53,000	$90,000	$250,000
Boat	20,000	28,000	50,000	100,000	200,000
Two buckets	3,600	4,400	5,200	9,000	20,000
Welder	1,000	1,000	1,000	1,000	1,000
Field shop	1,000	1,000	1,500	3,000	10,000
Bulldozer	7,000	10,000	10,000	10,000	10,000
Trucks and miscellaneous	2,000	2,000	3,000	5,000	10,000
Total	$59,600	$82,400	$123,700	$218,000	$501,000

Approximate costs of dragline dredge equipment for various units are given in Table 106. Diesel power is widely used in dragline operations principally because of the cost of bringing electrical power to the scene and because of the comparatively short life of the individual project. The portable character of the equipment makes it possible to arrange for a number of individual areas and then to work them in succession by moving the equipment, or as much of it as is economical to move, from one area to the next. To give an idea of the machinery costs, data from several sources, principally *Information Circular* 7013, *U.S. Bureau of Mines*, have been collected and presented in Table 107.

TABLE 107.—DIESEL EQUIPMENT USED IN PLACER PITS AND DRAGLINE-WASHER INSTALLATIONS—APPROXIMATE OR LIST PRICES

Equipment	Bucket capacity, cu. yd.				
	¾	1	1½	2	3
Diesel shovel:					
Recent list	$10,000	$16,000	$22,000		
1932 list	10,000	12,000	16,000		
Dragline:					
Boom length, ft	35	45	30	45	85
1932 list	$9,300	$11,000	$13,700	$17,250	$48,400
Washing plant	....	....	15,500	19,000	22,500
Wooden hull	....	....	1,600	1,800	1,850

NOTE.—List prices of diesel engines for 1935, based on continuous horsepower, were as follows: 95 hp., $3,950; 76 hp., $3,600; 59 hp., $2,737; 48 hp., $2,487; and 36 hp., $2,023.

List prices of diesel-electric generators were $5,516 for 60 kw., 3 phases, 60 cycles, 900 r.p.m.; $4,192 for 40 kw.; $3,198 for 25 kw.

List prices of diesel tread tractors for 1937 were $7,325 for 95 hp., with bulldozer $1,230 to $1,700; $5,300 for 60 hp., with bulldozer $1,040 to $1,260.

Small diesel power shovels are included, as they have been used in placer projects where stationary washing plants have been installed.

Excavator, Portable and Stationary Washing Plants.—The possibilities of excavation by power shovel and transportation by truck to a stationary washing plant have attracted placer operators at various times but the introduction of transportation units and the necessity of maintaining pit roads and ramps increase the costs greatly. Provision for dumping space is another difficulty. However, with rich enough gravel, say 50 cents per cubic yard, such equipment may be made to pay. The combination of stripping overburden by tractor-tread scrapers, power shovels or dragline excavators for gravel mining, and trucks for transportation is another that has been tried out and has some good points. Many such combinations are not successful either on account of low-grade gravel or too small a deposit to carry the capital expense for such equipment.

The most successful of dry-pit operations is the combination of semiportable sluice and dragline excavator. An installation of this kind makes use of a 100-ft. steel sluice with head box and inclined grizzly, mounted upon an 80-ft. light steel truss attached to heavy skid timbers, the height being sufficient to get dumping room at the side for the boulders and space for tailings at the end of the sluice. The prolong of the sluice is carried by a steel A frame and wire ropes.

A diesel-operated centrifugal is mounted beneath the head box and supplies water to a nozzle. This directs its stream from a position along the sluice into the head box, where the water churns up the gravel and disintegrates it, after which it flows down the sluices to the tailings sump. The water is returned to the pump through a slip-joint suction pipe. The washing unit is placed on one side of the area to be mined and a cut about 60 ft. in width is made by the dragline. When the cut is completed the sluice is moved upstream and a new cut is started. In wider deposits the sluice can be moved laterally instead of longitudinally. There are many variations of this combination. One employs two draglines, one for the excavation and the other to stack tailings over a wider area. An accessory is a heavy claw-shaped rake used to break up the bank and separate large boulders. This rake is used in place of the dragline bucket for preparing the ground for bucketing.

Dry Placering.—In deserts, where there is insufficient water for sluicing, alluvials are worked by a combination of portable screening units and dry washers, which effect concentration by transverse riffles set upon steeply inclined trays and winnowing by air currents supplied by bellows or fans. The final partially concentrated product is cleaned up by panning in water. Operations of this kind are on a very small scale and power is used only for screens and washers. The material treated in the air separators must be bone-dry.

Recoveries in Alluvial Mining.—Recovery by sluices, dredges, washing plants, or by other means is always less than 100 per cent. The distribution of gold in alluvial material is seldom uniform, nor is it consistently distributed either upon bedrock or within certain strata of gravel. The size of gold particles varies between wide limits, through flour, flaky, shot, or nugget. Attrition in excavation, screening, and washing may divide particles still further to some extent. Spillage, unless controlled, may result in losses of some magnitude; insufficient disintegration may send gold particles to the tailings pile; surges of water and packed riffles may sweep gold particles away; inefficient operation of machinery and recovery appliances may result in low recovery; bedrock may be cleaned carelessly; and, in dredging, corners and ridges may be left undredged. Recovery varies from day to day and from different parts of the deposit. Recovery of relatively coarse gold where disintegration is complete is very high, close to 100 per cent; recovery of finely divided gold is also high under similar conditions; recovery of flour and flaky gold is often very low, especially where clay is present in quantity. W. H. Lanagan gives an interesting account of the working of a very clayey placer at Pokrovsky, Siberia, in which the first working yielded 75 cents per cubic yard; after 2 or 3 years, a second working gave another 75 cents per cubic yard, and it was estimated that at least 75 cents per cubic yard still remained. Thus the first working gave only one-third extraction.

Tailings loss is very difficult to ascertain and to apportion, although many attempts have been made to sample coarse and fine tailings in dredge operation. The efficiency of sluices is determined by comparing the amount of gold caught in the upper sections with that found in tail sluices and by comparing successive undercurrents along the sluice line. Much attention has been given to the design, testing, and operation of distributors and tables. In some cases thorough disintegration may be the all-important factor. Generally a high recovery, 80 per cent or better, is made from the alluvial material delivered to the screens of the dredge where efficient operation prevails.

Drill Value and Dredge Recovery
(*C. W. Gardner*)

Drill value, cu. yd.	Acres	Per cent total acreage	Average value, cu. yd., cents		Gain, per cent	Loss, per cent
			Drill	Dredge, recovery		
Under 11 cents	27.9	0.7	9.21	8.60	....	6.6
Under 11 cents	1,638.1	43.8	7.68	9.34	21.6	
Bet. 11 and 12 cents	424.0	11.3	11.28	10.12	....	10.3
Bet. 11 and 12 cents	480.0	12.8	11.61	16.44	41.6	
Bet. 12 and 20 cents	582.5	15.6	16.92	13.10	....	22.6
Over 20 cents	392.0	10.5	45.86	20.93	....	54.4
Over 20 cents	198.5	5.3	33.15	35.97	8.5	
Totals	3,743.0	100.0				
Averages	......	.....	15.40	13.55	....	12.0

Note.—Gold at $20.67 per ounce.

H. Sawin, on the subject of quicksilver losses in gold dredges, gave the following examples. Yuba 20, with 18-cu. ft. buckets and 5,600-sq.ft. tables, required 2,200 lb. of quicksilver to charge tables at start; average quicksilver loss was 485 lb. per year (4 years) or 0.087 lb. per sq. ft. per year; digging rate averaged 10,000 cu. yd. per day, with 50 per cent passing over tables. Yuba 17, with 18-cu. ft. buckets and 8,335 sq. ft. of table area, required 4,000 lb. of quicksilver to charge tables and showed an average annual loss of quicksilver of 530 lb., or 0.064 lb. per sq. ft. per year. The rate of digging was 10,000 to 11,000 cu. yd. per day. Quicksilver losses were at the rate of 1 lb. per 10,000,000 lb. table material and were probably due to attrition. Bowie gives an example of quicksilver losses in hydraulic mining; the loss ranged from 4.4 to 6.1 per cent of the quicksilver used in the sluices.

Comparing dredge recoveries with estimates of gold content of the deposit as computed from drill-hole and shaft samples indicates close concordance in many instances and wide variations in others, ranging from lows of 20 to 50 per cent up to highs of 150 per cent or better of anticipated recovery. The chain between initial sampling and the delivery of the gold has many links. However, when a property has been sufficiently prospected by an experienced engineer, when the results have been intelligently interpreted, and when the calculations have been accurately made, the results obtained will indicate the gold content within reasonable limits. The table on page 491 gives a comparison between drill results and dredge recoveries.

Future of Alluvial Mining.—Although gold, platinum, and tin have been the principal objectives of alluvial mining, it is reasonable to expect that wider attention will be given to prospecting for chromite, gems, scheelite, and other industrial minerals. Close attention will be paid to the heavy accessory minerals and detailed examination of them will be made by spectroscope and chemical tests. Wider use will be made of low-cost excavating machinery for the removal of overburden. Washing plants will be better designed and will be more efficiently operated. Systematic sampling in some form may become common practice.

Reclaiming Dredged Land.—Successful reclaiming of dredged land has not been attained and partial success is usually at prohibitive cost. In backfilling, surface soil and fine material are deposited upon the bottom of the pond and the screen and stacker discharge forms an advancing bank on top at the rear. For partial reclamation it is necessary to invert the normal position of the fines on the bottom and the coarse gravel on the top. This was accomplished to some extent at Natoma by installing sand wheels to take the sluice discharge and dewater and deliver it to the stacker belt. In another method the sluice discharge was pumped to the stacker end and discharged over the gravel pile. Yuba provided a separate suction pump and discharge pipe to take the accumulation of sand from the pond bottom near the ladder end and deliver it to the top of the spoil pile. An attempt to reclaim dredged land was made at Natoma, but costs were very great and success uncertain.

In New Zealand and Australia advance stripping was practiced as early as 1905. J. H. W. McGeorge describes a later improvement of this method in 1943 as applied at Newstead, Victoria, Australia. The dredge was equipped with an overburden belt conveyor starting behind the ladder well and extending up a stacker boom at the rear of the dredge. While topsoil and barren overburden are being excavated, the bucket discharge is diverted to the belt conveyor and is delivered to the top of the spoil pile. While the deeper pay gravel is being dug, the screen discharge is dropped from a chute at the tail end of the hull and the table and slime discharge falls upon the top of the advancing coarse gravel bed. Land reclamation involved an additional dredging cost of about 1 cent per cubic yard. The cost per acre for reclaimed land was approximately $300. The depth of dredging averaged 24.4 ft.

Drift Mining.—Underground methods applied to alluvial deposits are included under the term "drift mining." As such methods are more expensive than surface operations, the deposits must be richer. Usually drift mining is attempted only when the pay channel is concentrated and too deep to be mined by open-pit methods. Gravel grades are about $4 per cubic yard and upward. Thickness ranges from 4 to 12 ft. or more; width is from 20 to 200 ft. or more; depth below the surface may be 30 to 200 ft. or more and the length may be up to a mile or more. Much water is usually present, and in Alaska frozen gravel may sometimes be encountered.

Where an adit is possible, it is driven on a drainage grade, ¼ to ½ per cent on the shortest practical course to get beneath the deepest part of the channel. Where a shaft is used, it likewise is placed so as to command the lowest part of the channel. Where the channel grade is flat the shaft may be sunk in the mid-portion of the channel length. In the Fairbanks district, Alaska, 6- by 6-ft. cribbed shafts were sunk in the middle of a 500-ft. block of pay channel averaging 100 ft. in width. Surface conditions are a factor in selecting shaft sites. Predraining of the channel by boreholes and deep-well pumps facilitates shaft and other development work. The shaft bottom is sumped and is the drainage focus of the mine.

From the shaft or adit a main drift is extended the length of the claim or pay streak. The drift is 5 by 7 ft. or 6 by 6 ft. and timbered with sets from 4 to 5 ft. on centers. Lagging is placed on top and sides. Where the gravel is free, forepoling is necessary. At intervals of 100 ft. or more, crosscuts are extended from the main drift and the width of the pay streak is determined. Mining begins at the end of the main drift. The working faces are at right angles to the drift or across the pay streak. The first working face is obtained by driving crosscuts on either side of the main drift to the edges of the pay streak. In moderately cemented gravel, posts, caps, and lagging, or props and headboards, are required. In loose gravel, crosscuts are driven by forepoling. From the side of the crosscut a slice is taken off, 5 or 6 ft. wide and the length of the crosscut. This is timbered with props and headboards or forepoled. Boulders and large stones are packed back in the crosscut. Successive slices are taken off in this manner until the shaft or adit is reached. In ground permanently frozen, steam points 5 to 8 ft. long are driven into the faces and the gravel is thawed before excavation. Very little timber is required in frozen ground. Props and pillars are used where the roof slabs badly.

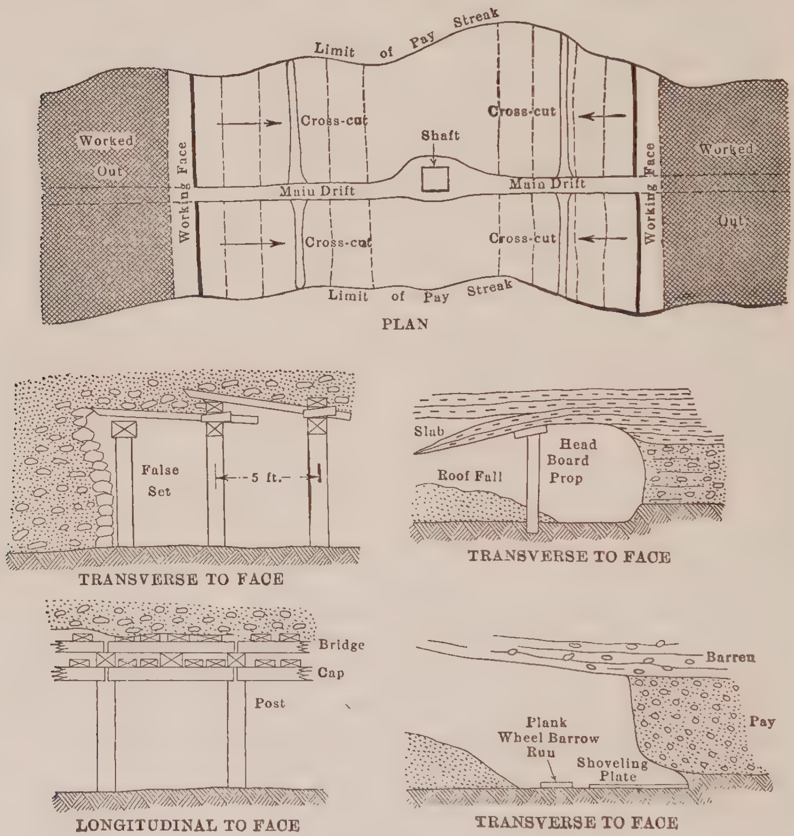


Fig. 229.—Drift mining.

Transportation is effected in main drifts by track and car, but storage-battery locomotives may be used where tramming distances are excessive. Wheelbarrows are often used in smaller operations. Mechanical loaders may be used in large operations. Both skip and cage hoisting may be used in shaft mines. In some of the Alaskan drift mines a bucket with carrier and trolley cable was used for hoisting and delivering to the dump box of the sluice. Steam or gasoline hoists are used, also electric hoists where power is convenient. In some mines where the gravel is cemented, compressed-air drills are used. Very long adits are equipped for electrical haulage. The plan and cross-sections of workings are shown in Fig. 229.

Ventilation is not a serious problem in small mines, but an air pipe and power-driven fan are necessary where the adit is long. Mines opened by a shaft are ventilated by an auxiliary shaft where the workings are extensive. The auxiliary shaft serves also as a working shaft.

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Certain types of ore deposits such as flat bodies of iron ore, lenses and sheets of lead and zinc ore, and the porphyry copper deposits admit of preliminary exploration by boring. By this method practically all the information necessary for the planning of the entire mining work can be accurately determined. The method is both rapid and economical, and where conditions admit of its use it should not be neglected. Bedded deposits such as coal, salt, gypsum, and the like can be as satisfactorily explored by boring as the foregoing. Precious-metal deposits, excepting placers, and deposits of the metals where they exist as narrow veins cannot as a rule be explored by boring, and underground workings are best. A preliminary geological study is always essential; in most cases it will conclusively indicate the best method for the preliminary development and in some instances the final development method.

Development workings consisting of shafts, drifts, crosscuts, raises, etc., are generally driven in ore and where this is the case the development is called "productive development." More or less work is, however, driven in the country rock, and to this the term "dead work" is applied. From economic considerations the minimum amount of dead work should rule. Development workings cost more than stoping per unit excavated, and for this reason development should be planned to render accessible the maximum quantity of ore for a minimum volume of development workings. Where this principle is followed a considerable saving in initial capital results.

NOMENCLATURE

Terms Peculiar to Metal Mining.—The following terms, in current use, are defined for the purpose of accurate usage.

Shaft.—A vertical excavation of restricted cross section and relatively great depth, used for access and working.

Incline.—An excavation of the same nature as a shaft and used for the same purposes but driven at an angle from the vertical.

Raise.—An excavation of restricted cross section, driven vertically or at an angle upward from a drift and in the ore body. It is used as a manway, timber chute, waste chute, ore chute, or for ventilation.

Winze.—An excavation of restricted cross section driven vertically or at an angle downward from a drift and in the ore body. It is used as a manway, timber chute, waste chute, ore chute, or for ventilation.

Underground Shaft or Incline.—A shaft or incline that is driven from underground workings and is not in the vein or ore body. The term "internal shaft" has the same significance.

Adit.—A horizontal gallery driven from the surface and giving access to an ore body that is worked through it; used sometimes solely for drainage or ventilation or both. The term "tunnel" is frequently used in place of "adit" and has the same signification.

Drift.—A horizontal gallery driven along the course of a vein. It is called a "footwall drift" when driven in the footwall, and a "hanging-wall drift" when driven in the hanging wall.

Crosscut.—A horizontal gallery driven at right angles to the strike of a vein. When driven at an angle to the vein and across it the same term is applied.

Level.—All the horizontal workings tributary to a given shaft station are collectivelly called a "level." Levels are designated as 100 ft., 200 ft., etc., according the vertical depth from the surface.

Main Level.—Where the ore mined from several levels is hauled upon one level to the shaft, this level is designated as the "main level." The term "main haulage level" is used with the same signification.

Sublevel, Intermediate Level.—A level driven from a raise or manway and not connecting directly with the working shaft is termed an "intermediate level" or "sublevel."

Transfer.—A vertical or inclined connection between two or more levels, used for an ore pass.

Terms Peculiar to Coal Mines. *Shaft, Air Shaft, Main Working Shaft.*—The significance of these terms is the same as the equivalent terms used in metal mining.

Main Entry.—The principal horizontal gallery giving access to a coal seam and used for ventilation, haulage, etc. Where two entries are driven in parallel the term "double entry" is used. With three parallel entries in juxtaposition the term "triple entry" is used.

Cross Entry.—A horizontal gallery driven at an angle or at right angles to the main entry is termed a "cross entry." The meanings of "double cross entry" and "triple cross entry" are easily understood from the preceding definition.

Butt Entry.—A horizontal gallery driven parallel with the main cleat of the coal seam.

Face Entry.—A horizontal gallery driven at right angles to the main cleat of the coal seam.

Gate.—A horizontal gallery giving access to a working face in the long-wall method of coal mining.

Mother Gate.—A horizontal gallery to which the gates are tributary.

Level, Lift.—Terms used to designate the working entries in a coal seam that dips at an angle.

Slope.—The main working gallery or entry of a coal seam which dips at an angle and in which trains of mine cars are hauled.

DEVELOPMENT WORKINGS

Main Opening.—The principal working opening of a mine may be an adit, entry, slope, incline, or shaft. Good mining practice requires the construction and maintenance of at least another opening, which

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DEVELOPMENT WORKINGS

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If we assume a narrow vein dipping at a steep angle and a surface topography of low relief, a shaft is the most suitable method of opening. It may be vertical as shown in Fig. 230*a*, an incline in the plane of the vein as in Fig. 230*b*, or an incline in the footwall as in Fig. 230*c*. The vertical shaft gives the greatest vertical depth for the least footage but requires the driving of long crosscuts to the vein at points above and below the intersection of shaft and vein. Excessive footage can, however, be avoided by workings in the plane of the vein, tributary to a main level, which is placed at the intersection. The principal objections to the vertical shaft are its position in the hanging wall, which is often unstable, the long crosscuts, and the difficulty of maintenance at the intersection of the vein and shaft. The necessity of locating the shaft mouth at a site suitable for the surface plant and for convenient access often overbalances the foregoing objections. The combination of a vertical shaft to the vein with an incline from the intersection to greater depth avoids the use of crosscuts but sacrifices simplicity in the hoisting arrangements.

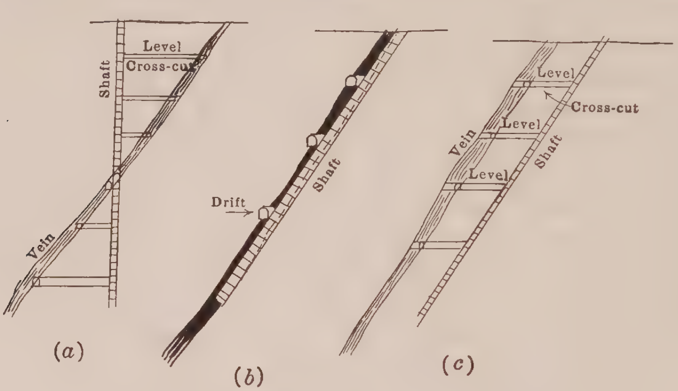

Fig. 230.—Position of working shaft relative to vein.

The sinking of an incline in the plane of the vein has the disadvantage of increased maintenance where the vein is heavy. A shaft pillar must also be left on either side to protect the shaft. In good ground an incline in the vein develops the maximum length of vein for a given footage and is probably the best method if a suitable shaft site can be found.

The position of the incline in the footwall has the advantages of maximum stability if the footwall is solid, minimum maintenance cost, and the avoidance of shaft pillars. On the whole, greater permanency is obtained with an incline in this position.

The important factors in determining the position and type of opening are a convenient and safe site for the surface plant, minimum footage, low maintenance costs, the physical nature of the hanging wall, vein, and footwall, and the dip of the vein.

If the topography is of high relief, the vein may be best opened by an adit. This should start from such a position as to give sufficient vertical depth without excessive length. The possibilities of the situation can be determined only by careful surveys. The adit has the advantages of lower cost of construction and a minimum of hoisting; it also provides for drainage. A comparison of costs of construction and maintenance and estimates of the cost of providing access by roads or other means of surface transportation, costs of operation, and of the permanency of the workings must be made where there is a question of the suitability of an adit as compared to a shaft.

A flat coal seam at depth is best opened up by a shaft. Two shafts are invariably driven, one serving as an upcast and the other as a downcast and for working purposes. They are placed in the most convenient position for the surface plant and at a distance of from 100 to 150 ft. apart. (If the seam outcrops, it is opened up by a pair of entries driven at right angles to the main cleat from the most convenient surface site on the outcrop.) If the seam is inclined, the entries are driven down the dip. Under certain topographic conditions crosscuts are driven from the most convenient plant site until they intersect the seam, where slopes following the dip are constructed.

Thick ore bodies of great lateral extent are best opened up by shafts sunk in the country rock close to the deposit. Where topographic conditions are suitable an adit can be used.

Subordinate Workings.—The method of mining as well as the size and position of the deposit determine the nature of the subordinate development workings that are connected to the main working openings. There are certain similarities in most development plans. Whether the deposit is a thin sheet, flat, or inclined, it is divided by the development into rectangular blocks, each block constituting a unit that is subsequently removed in the mining operations. If the deposit is thick it is divided into a number of horizontal slices, each slice being in turn divided into rectangular blocks. Certain well-recognized methods have been evolved in mining practice, and the development characteristic of each will be described.

Development for Underhand and Overhand Stoping.—From the shaft stations or crosscuts drifts are driven along the vein in both directions. The vertical distance between the drifts ranges from 50 to 300 ft. The usual distance is 100 ft. Where ore continuity is very uncertain, drifts are closely spaced; but where continuity is well established, the greater the spacing between the levels, the lower the cost for development and the smaller the capital tied up in the equipment of the levels. Each level is provided with a station, facilities for loading at the station, and a track system for transportation. It is obvious that the fewer the levels for a given ore body, the less expensive will this equipment be. Raises and winzes are extended from the drifts at intervals. Raises are generally used, since they can be more economically driven than winzes. Both are used to establish the continuity of the ore body between levels. They also serve to establish connections for ventilation, and stoping can start in either direction from a raise. Where continuity is established, raises are driven at intervals of from 200 to 300 ft. along the drift. The block of ore thus developed extends between the drifts and is of a length varying from 200 to 300 ft. No regular spacing is possible with small ore bodies erratically distributed along a drift. Winzes are sunk below a given level in order to test out the ore possibilities below the level before another level is opened out. From them drifts are sometimes driven, and the extent of the ore body at a given level is established in advance of further shaft sinking. The principle of least work

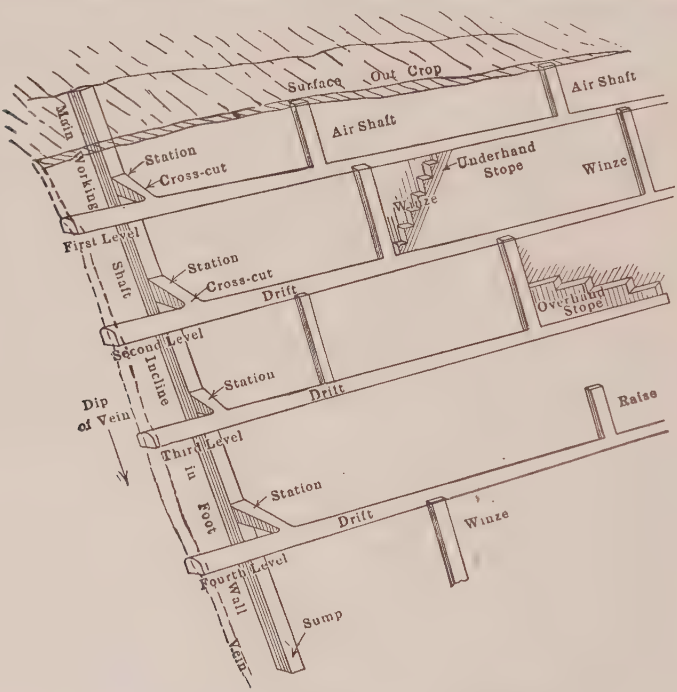

FIG. 231.—Layout of development on an inclined vein.

is observed by the miner in this feeling-out process. Where continuity is established, the opening out of the lower level usually follows, since it is more economical to stope from a lower level than otherwise. Figure 231 illustrates the development workings of a vein in relation to the main opening. The connections to the surface for ventilation and egress are also shown.

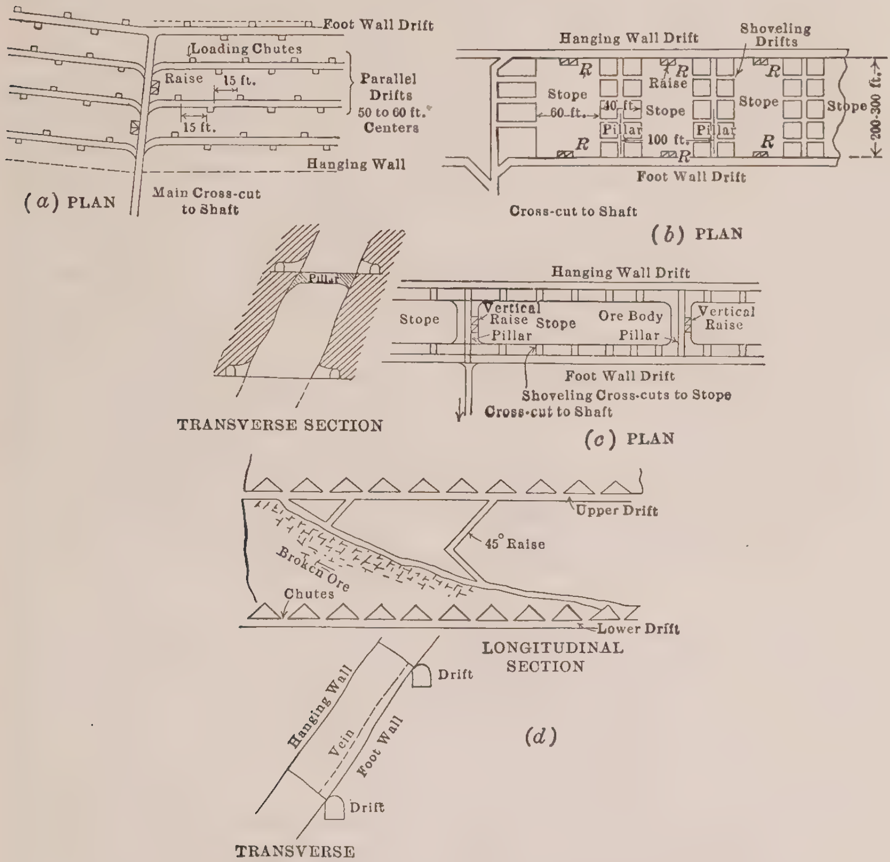

Fig. 232.—Development for shrinkage stoping.

Development for Square-set Mining.—The method of development described in the preceding is applicable to ore bodies from 1 to 8 ft. in width. For wider veins the same general plan is followed. It is necessary, however, to drive crosscuts at intervals of 50 to 100 ft. in order to determine the thickness of the vein and the position of the walls. The drift can be driven near the hanging, in the middle, or along the footwall. The last is the preferable position. Drifts of larger section are driven for wide veins than for narrow veins. Where square-set mining is to be used, the only essential difference is the use of sill-floor sets for timbering the drift. These necessitate the excavation of a section 10 ft. high by 7 ft. wide. Where the vein is irregular in its course, a small drift is best, and the position of the wall is determined from this in order to lay out the sill-floor plan more accurately. Raises are driven at intervals of 50 to 100 ft. and are usually timbered with square sets, the raise being two sets wide. As with the drifts preliminary raises can be driven and supported by stulls.

Development for Shrinkage Stoping.—The development methods for shrinkage stoping are essentially the same as those just described. Where continuity is well established, however, levels are driven at intervals of 150 to 175 ft. With wide veins, parallel drifts at 40- to 50-ft. centers are driven from a crosscut extending from wall to wall, as shown in Fig. 232*a*. Figure 232*b* illustrates a plan of the development used on a wide lode at the Homestake mine. Footwall and hanging-wall drifts are driven and crosscuts connecting them are spaced at 100-ft. centers. From the crosscuts short drifts are driven to the stopes and from the ends of these the stope is opened out transversely to the strike of the lode. In Fig. 232*c* the same method is applied to a narrower lode. Here the stopes are placed along the strike, and the short crosscuts extending from the footwall and hanging-wall drifts are extended to the lode. In both cases raises are extended at either the ends or the sides of the stopes. Figure 232*d* illustrates the method used at the Melones mine, where the vein dips at an angle of 60 to 70 deg. The drifts are driven in the footwall in proximity to the ore body, and raises are extended upward at a 45-deg. pitch.

Development for Top Slicing and Cover Caving.—The main level of a Mesabi iron mine is shown in Fig. 233. Two parallel drifts from 66 to 100 ft. apart are driven at the bottom of the ore body, parallel to the axis of the deposit. From these drifts crosscuts are extended at 50-ft. intervals to the outer limits of the ore body. The crosscuts connect with the main drifts by curved galleries. Raises are extended vertically upward at intervals of 50 ft. until they are close to the upper horizon of the ore body. The first sublevel is extended from the tops of these raises, which terminate on the floor of the first slice to be removed.

The sublevel development consists of a series of parallel crosscuts directly above the main crosscuts. They are extended to the periphery of the ore body or to the boundaries of the property. Narrow drifts connect the sublevel crosscuts at intervals of 100 ft. or more. The appearance of the sublevel is shown in Fig. 234. One or two timber shafts are sunk to the first sublevel to afford ventilation and access and to enable the timber to be delivered to the working places without hoisting. The raises serve as ore chutes, a few at convenient points being used for manways.

Mining begins on the uppermost sublevel, and sublevels are driven below in sequence from the top down. The thickness of the slice mined determines the vertical interval between sublevels. It is from 10 to 15 ft.

The method described above is sometimes modified by omitting the

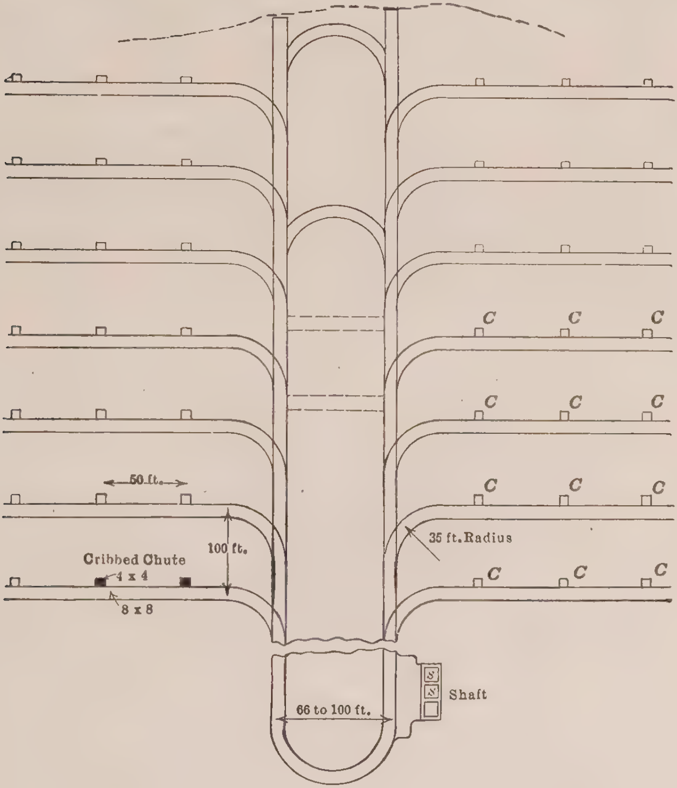

FIG. 233.—Main level plan, development for top slicing.

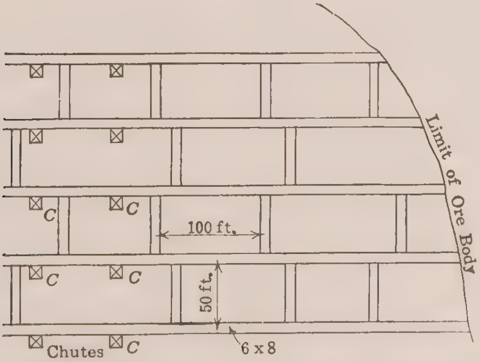

FIG. 234.—Sublevel plan, development for top slicing.

crosscuts on the main level and restricting the development of this level to the two parallel drifts and to the manways and chutes spaced at 50-ft. intervals along both drifts. The sublevels are opened out in the same way as described before. It is evident that the latter plan is somewhat lower in first cost, but the expense for tramming on the sublevels is increased by the greater distance from working chambers to chutes. A relatively narrow and thick ore body is best developed by the latter method and thin, wide ore bodies by the former. The use of the scraper, mechanical loader, and shaking-pan conveyor enables a greater spacing of transfers or chutes.

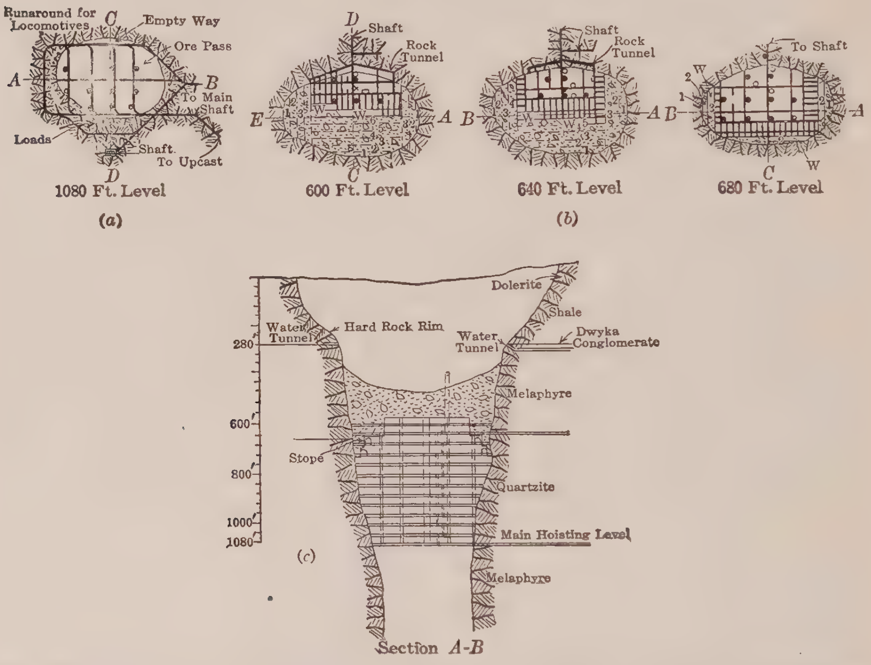

FIG. 235.—Kimberley method of development for combined top slicing and shrinkage stoping. (*Eng. Min. Jour.*)

Development for Combined Top Slicing and Shrinkage Stoping.—The practice at the Kimberley diamond mines is taken to illustrate the method. Figure 235*a* shows the main-level development and the connecting ore passes or chutes. Main levels are established at 500-ft. intervals. Sublevels are driven at 40-ft. intervals and the sublevel workings at a unit distance of 22.5 ft. apart or some multiple of this distance. The plans of several sublevels are shown in Fig. 235*b*. It should be noted that the driving of the working drifts and crosscuts at the closest interval (22.5-ft. centers) is done only close to the stopes.

As the slice is drawn back the larger blocks are divided. The vertical section, *c*, shows the different sublevels in relation to the main level.

Development for Top Slicing and Partial Ore Caving.—The practice of the Newport mine on the Gogebic range is taken to illustrate this method. Main-level development is shown in Fig. 236. Sublevels are placed at intervals of 15 ft. The plan of a sublevel is shown in Fig. 237. Connection between main and sublevels is through raises, some of which serve as manways and others as ore passes.

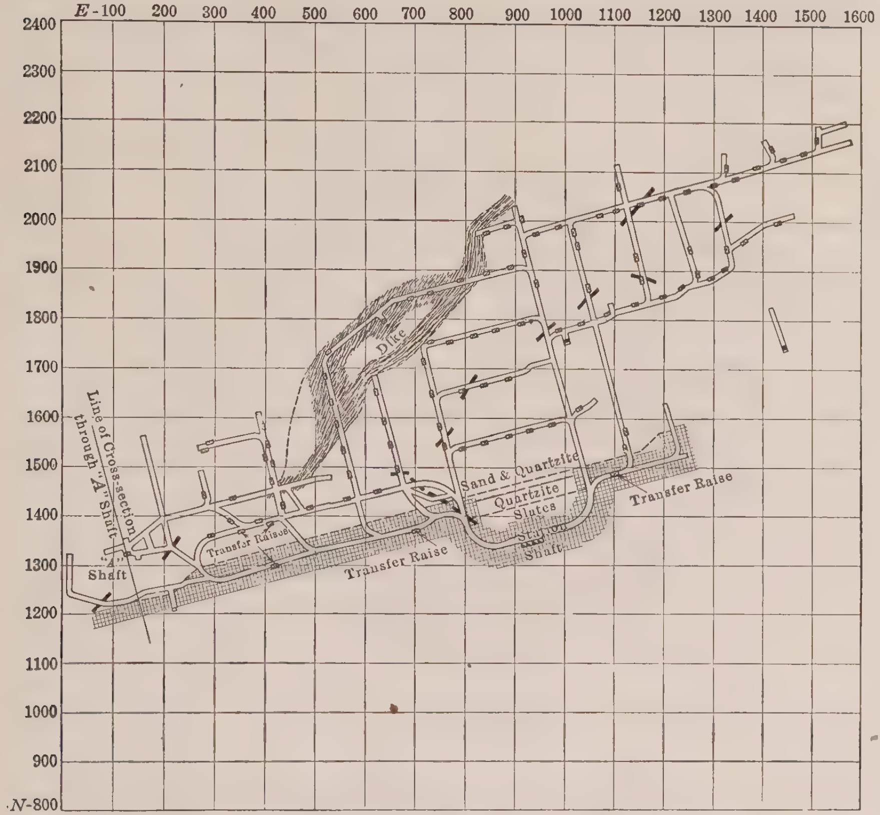

FIG. 236.—Plan of main level, development for combined top slicing and ore caving. (*Trans. A.I.M.E.*)

Development for Chute Caving.—The Pioneer mine on the Vermilion Range, Minn., affords a typical example of this method of development. The levels are placed at 100-ft. intervals, and drifts and crosscuts divide the area of the ore body into blocks approximately 100 ft. square. The level is laid out for power haulage. As far as is practicable each level is laid out to conform to the level above, and the workings are positioned immediately above the corresponding workings on the level below. From the level drifts and crosscuts, vertical raises are extended at intervals of 25 ft. From the raises and between two levels, two sublevels are extended at vertical distances apart of 33⅓ ft. The sublevels consist of drifts and crosscuts and conform as far as practicable with the layout of the level. The blocks on each sublevel are subdivided still further into pillars approximately 25 ft. square. As soon as the uppermost sublevel has reached the boundary of the deposit caving can begin, presuming that the level above has been worked out and that the timber mat and caved capping are down.

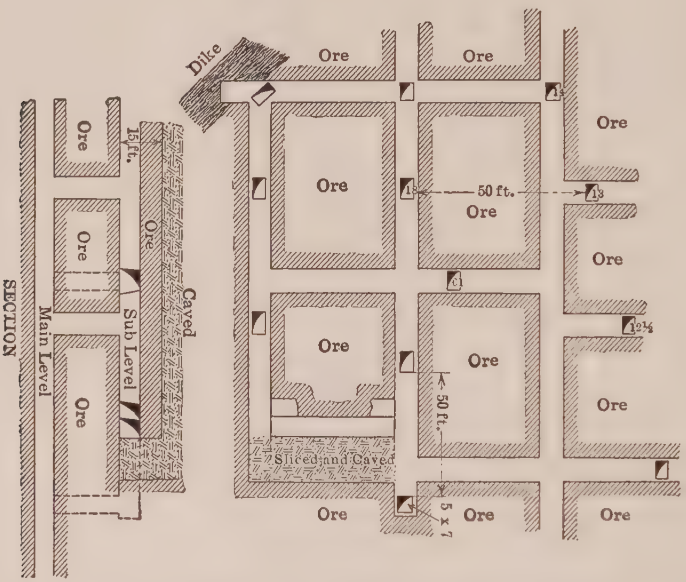

FIG. 237.—Plan of sublevel development for combined top slicing and ore caving. (*Trans. A.I.M.E.*)

Development for Block-caving.—Block-caving as practiced at the Tobin mine, Crystal Falls, Mich., affords a historic example.[1] The main levels are placed at vertical intervals of 125 ft. From the shaft station a crosscut gives access to the ore body. A main drift is driven from the crosscut along the hanging wall, and from this crosscuts are extended to the footwall at 24-ft. centers. The crosscuts are joined by a narrow drift that follows the footwall. Along the crosscuts, at intervals of 15 ft. and placed alternately right and left, short raises are extended to the sublevel that is to be opened out 25 ft. vertically above the main level. The sublevel is then opened out by drifts and crosscuts, which divide the sublevel area into pillars 10 by 16 ft. and connect the tops of the raises. The tops of the raises are made funnel-shaped. At the ends of the block two additional crosscuts are driven at vertical intervals of 25 ft. above the first sublevel crosscut. Raises at intervals of 30 ft. connect the three end crosscuts, which are superimposed one above the other. They are extended 12.5 ft. above the uppermost crosscut. The purpose of the end crosscuts and raises is to weaken the ore block on the ends of the block which it is proposed to cave. Figure 238*a*, *b*, and *c* illustrates the method.

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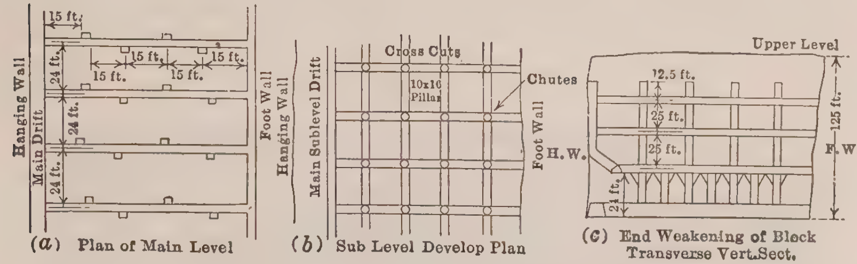
[1] *Trans. Lake Superior Min. Inst.*, vol. 16, p. 218.



(*a*) Plan of Main Level (*b*) Sub Level Develop Plan (*c*) End Weakening of Block Transverse Vert. Sect.

FIG. 238.—Development for block-caving. (*Trans. L.S.M.I.*)

At the Ohio mine (Utah), an early example of block-caving, a different system of development was used. The levels are at 100, 300, 400, 500, and 750 ft. The sublevel interval was 30 ft. and later 60 ft. On the sublevel at the bottom of the blocks the drifts and crosscuts divide the ore into blocks 20 by 50 ft., measured from center to center of the drifts and crosscuts. The block mined as a unit is 100 by 100 ft. square and 60 ft. thick. In place of vertical raises, as many as 150 inclined raises on an angle of 45 deg. are driven, a raise to 66 sq. ft. of bottom area being used to a unit. The inclined raises or chutes are constructed in the 60-ft. block immediately below the sublevel on which the undermining of the block above is to take place. The chutes are connected to branch chutes and these in turn to main chutes, which connect with the main adit level. The sketch shown in Fig. 239 will make the arrangement clear. The blocks are not weakened at the ends. The foregoing description is valuable historically. Later examples are described under mining methods.

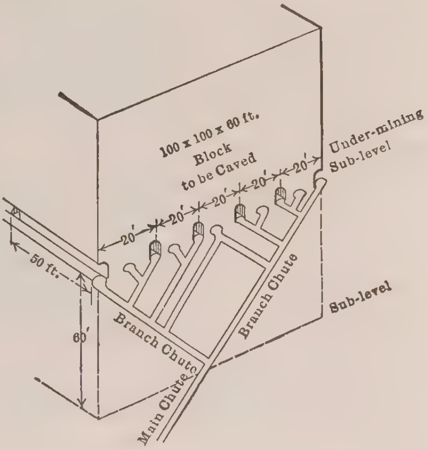


FIG. 239.—Block-caving at the Ohio mine.

Development for Longwall Mining.—The development required for a long-wall mine is comparatively simple. The working shafts and air shafts are first sunk, and from the working shaft a main haulageway is extended in both directions on the strike of the seam. From the air shaft an airway is driven at right angles to the main haulageway, at the intersection of which an overcast is constructed. Both workings are driven to the limits of the shaft pillar that is to be left as a protection to the shaft. The ends of the air- and haulageways are connected by a pair of parallel entries driven about the shaft pillar. A pillar 15 ft. in thickness is left between the entries. As soon as the entries are completed the narrow pillar between and a 15-ft. slice from the edge of the shaft pillar are mined out and the space is packed solidly with waste, openings being left at intervals. The mine is then ready for operation. Figure 240 shows the plan of the workings.[1] The method is used for long-wall advancing. For long-wall retreating, haulageways and airways are driven in pairs to the boundaries of the property and are then connected by a double entry, from which mining on the retreating system can be started.

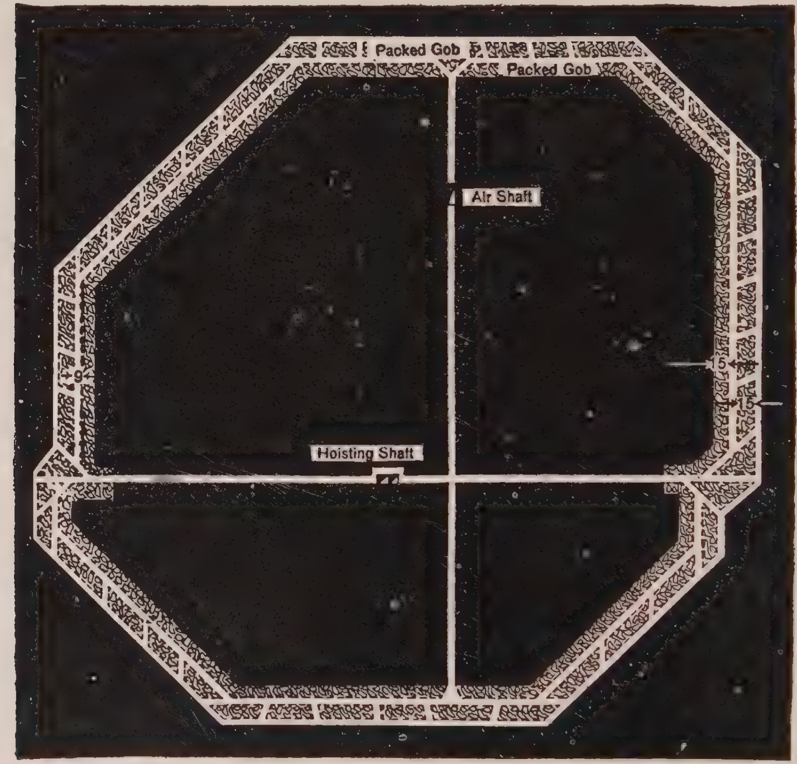


Fig. 240.—Plan of long-wall development. (*Illinois Coal Mining Investigations.*)

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[1] Taken from *Bull.* 5, *Illinois Coal Mining Investigations*, p. 14.

Figure 241 shows the system of entries subsequently driven and a somewhat different system of starting the long-wall advancing system.[1]

Development for Room-and-pillar Mining.—The simplest method is known as the double-entry system. Two main entries are driven at right angles to the cleat with a pillar or rib 50 ft. in thickness between. Break-throughs are spaced at 75-ft. intervals. At distances varying from 150 to 600 ft. butt or side entries are driven, and from them the rooms are turned off. Where the seam pitches, the main entries are driven on the pitch and the side entries on the strike of the seam, although this plan is sometimes reversed where the pitch is more than nominal.

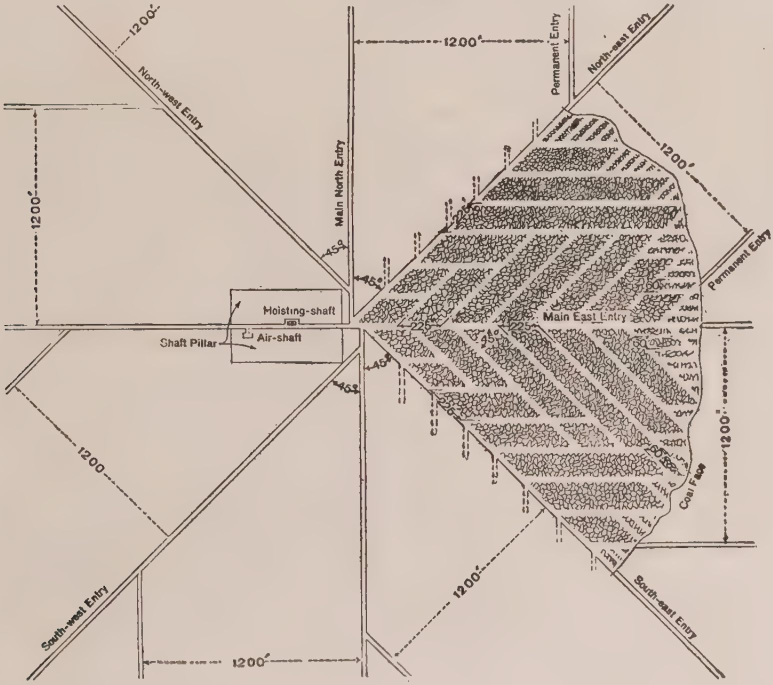

FIG. 241.—Plan of long-wall development. (*Illinois Coal Mining Investigations.*)

Figure 242 illustrates a method of development used by the Monongahela River Consolidated Coal and Coke Co. in the Pittsburgh region. The room lengths are 250 and 230 ft. There are four main or face entries and two butt entries, 480 ft. apart. The method is applicable to a flat seam.[2]

Development for Bord-and-pillar Working.—Main entries are driven in double pairs, butt or side entries in pairs. The bords, driven face on, are narrow rooms 18 ft. or less in width. The "walls" are driven at right

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[1] *Ibid.*, p. 18.

[2] *Trans. A.I.M.E.*, vol. 41, p. 233.

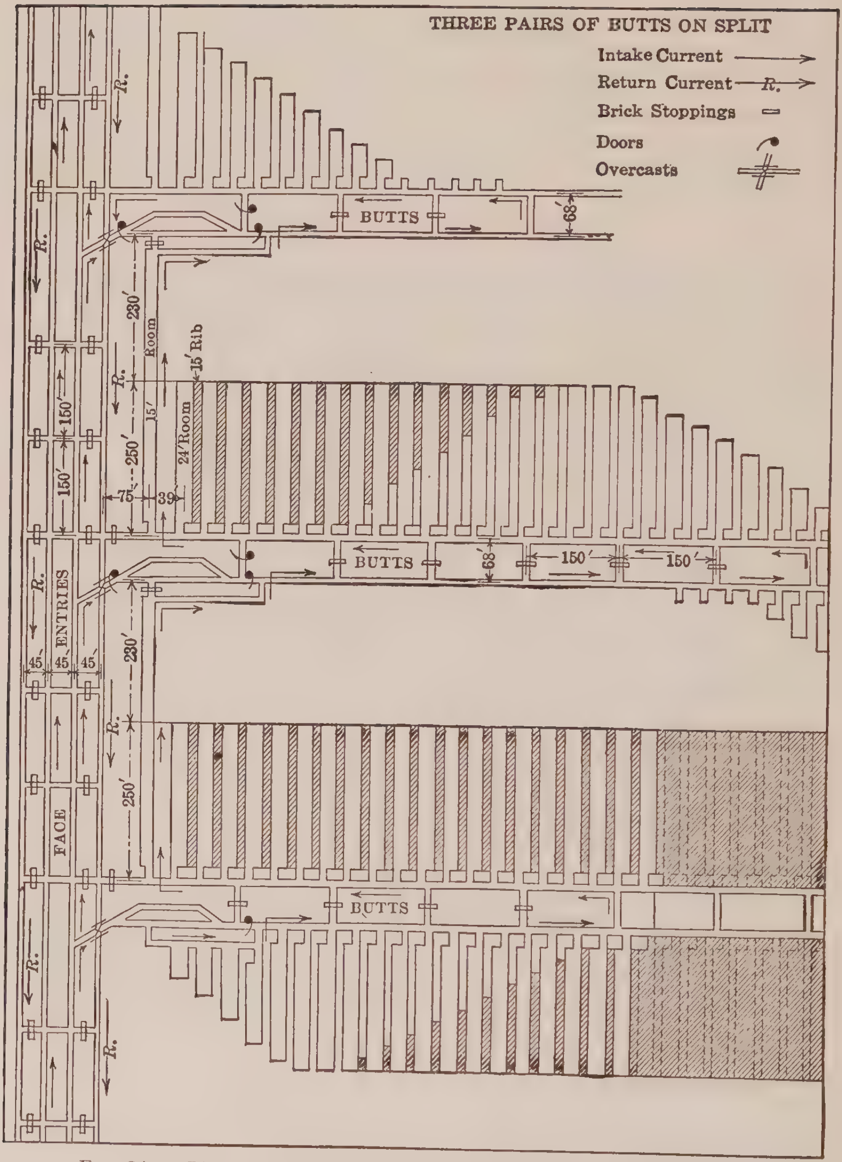

Fig. 242.—Plan of room-and-pillar development. (*Trans. A.I.M.E.*)

angles to the bords and are usually narrow, 9 ft. wide. The result is to divide the coal into a series of large square or rectangular pillars. Each pillar is a unit in the subsequent mining operations. The system as applied by the Thompson-Connersville Coke Co. is shown in Fig. 243.[1]

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[1] *Ibid.*, p. 231.

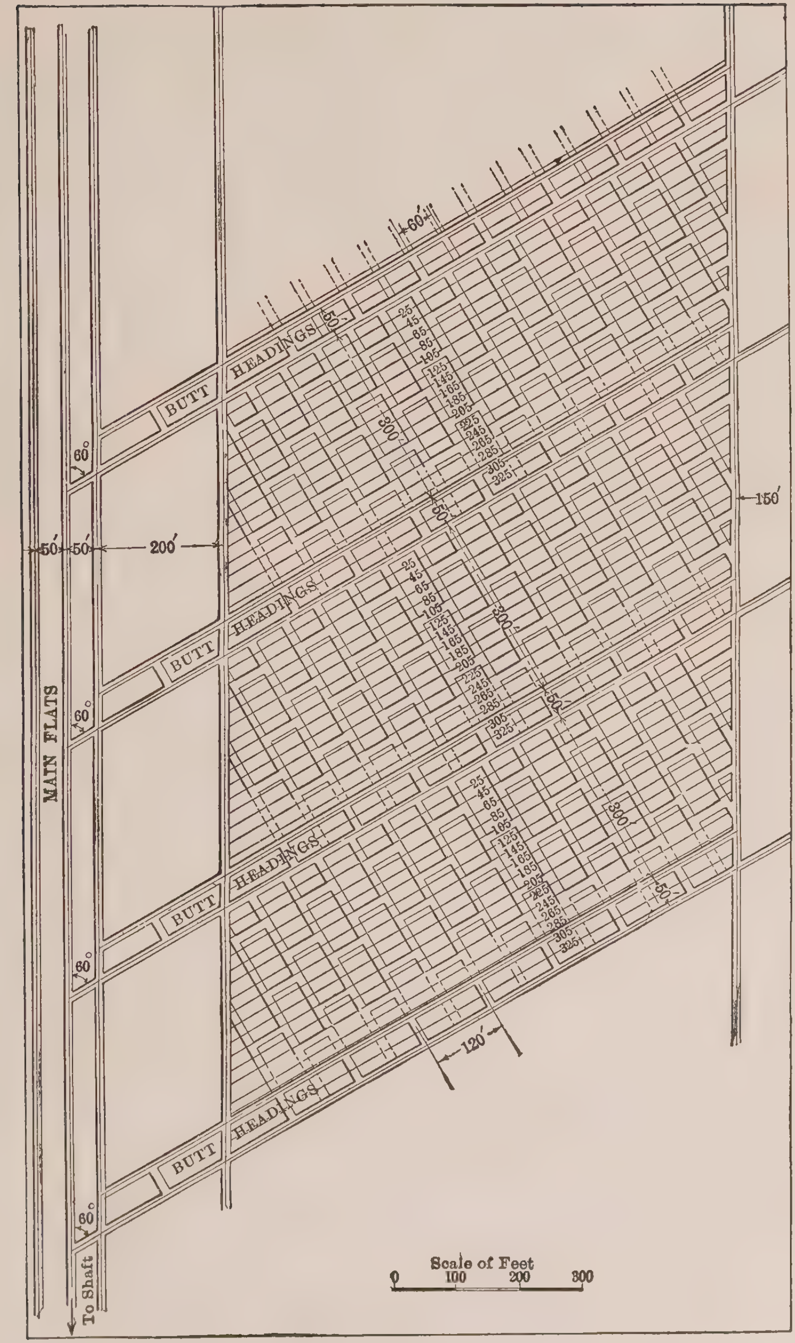

FIG. 243.—Plan of bord-and-pillar development. (*Trans. A.I.M.E.*)

DIMENSIONS OF DEVELOPMENT WORKINGS

Shafts.—Shaft sections are rectangular, square, elliptical, or circular. The rectangular section is under most circumstances the most convenient. It is not suitable where the ground surrounding the shaft section is heavy, as where soft sedimentaries are penetrated. Under such conditions elliptical or circular sections are best. Custom plays a certain part in the choice of the section. For example, in England and Germany shafts of circular section are the rule, while in America and South Africa the rectangular section is almost exclusively used. In most metal-mining districts where hard rock prevails, the rectangular section is favored; while in colliery districts the circular section is more apt to be used. There is an increasing tendency for metal miners to use the circular shaft.

Shaft sections are given usually as the inner dimensions of the compartments, the clear dimensions within the shaft timbers, or the dimensions outside the shaft timbers. For rectangular-shaft sections the size of compartments varies; the following are some of the dimensions in use.

For skip-and-cage compartments	4 × 4, 4 × 4.5, 4 × 5, 4.5 × 5, 5 × 6
For pump-and-manway compartments	4 × 5, 4.5 × 5, 5 × 5, 5 × 6, 6 × 6
For large cage compartments	5 × 10, 6 × 11

Shaft sizes are illustrated in the following:

Anaconda, Butte, Mont.	6.75 × 20.3 ft. over-all
Tamarack, Mich.	8.83 × 29.16 ft. over-all
Red Jacket, Mich.	15.5 × 25 ft. over-all
Pennsylvania bituminous coal mines	12.33 × 25.5

Some South African shafts have the following dimensions:

No. 4 shaft, Spring Mines, Ltd., inside dimensions 28 by 10 ft., divided into five compartments, 10 by 5 ft.; timber sets, 9- by 9-in. wall plates; 8- by 8-in. studdles; sets 7½-ft. centers; guides 7¾ by 3¾ in.; hanging bolts 10 per set, 1⅛-in. diameter; depth 4,023 ft.

South shaft, City Deep, Ltd., concrete rings, 20 ft. inside diameter, 10 ft. apart, 18 by 15 in. in section with 24-in. steel pegs; depth 4,500 ft.; two-cage compartments, cages 3½ by 15½ ft., pipe compartment and cable compartment; guides 80-lb. steel rails, standard section and flat bottom.

Other shafts are as follows: No. 3 shaft, Frood mine, 28 ft. 2 in. by 16 ft. outside timbers, 2,800 ft. in depth; 12- by 12-in. timbers; divided into four main compartments, 14 ft. 4 in. by 6 ft. clear, No. 1 compartment being divided into a skipway 6 by 6 ft., ladderway 3 by 6 ft., and a pipeway 4 ft. 8 in. by 6 ft.; No. 2 compartment into a skipway 7 ft. 2 in. by 6 ft. and service cageway 6 ft. 8 in. by 6 ft.; and No. 3 and 4 compartments, 14 ft. 4 in. by 6 ft. for material cageways. Newport shaft, excavation 15 by 23 ft., divided into two skipways, 5 ft. 9½ in. by 6 ft. ½ in., two cageways 5 ft. 9½ in. by 10 ft., pipe- and ladderways each 5 ft. 9½ in. by 3 ft. ½ in.; supported by steel sets, 6-in. H sections on 8-ft. centers.

The largest shaft section that I have seen described is that of a colliery at Carbondale, Pa., 12 by 52 ft. Circular shafts are 9, 10, 12, 15, 16, 18, 19, 20, and 21 ft. in diameter. Probably the most useful compartment size for rectangular shafts is 4.5 by 5 ft. in the clear for skips or cages, 5 by 6 for pump and manway, and 15 ft. diameter in the clear for circular shafts. Two cage or skip compartments and one manway are sufficient for mines of moderate to large output. An additional compartment for a cage is required where the shaft is to be of large capacity. For very deep mines, two skip and two cage compartments and a manway compartment are required. A comparison of circular and rectangular shafts is given in Table 108.

Table 108.—Comparison of Rectangular and Circular Shafts

	Rectangular	Circular
Skip compartments	4.5 × 6 ft.	4 ft. 7 in. × 5 ft.
Number of skip compartments	2	2
Cage compartment	6 × 10 ft.	6 × 10
Ladder compartment	4 × 6 ft.	5.5 × 4
Inside dimensions	25.5 × 6 ft.	17 ft. dia.
Outside dimensions	27.5 × 8 ft.	20 ft. dia.
Inside area, square feet	153	227
Outside area, square feet (area excavation)	264	314
Area of compartments, square feet, or used area	104	139
as per cent of area of excavation		
Inside area	57	72 (60)[a]
Area compartments	39	44 (37)[a]
Area lining or timbered space	43	28 (40)[a]
Waste area	16	28 (23)[a]

[a] Allowing 5 per cent for overbreaking in case of circular shaft.

The arrangement of the shaft compartments is illustrated in Fig. 244. Eight plans are shown. The selection of the size, number, and arrangement of the shaft compartments is determined by the future or present working requirements of the mine in question. Without knowledge of the size of the deposit and its value and, from these factors, the probable daily output, the engineer is very much in the dark in planning shaft facilities. Most large mines, however, pass through a number of stages before their possibilities in respect to maximum production are realized. Each stage is marked by the discarding of old shafts and the construction of new. The final improvements are thus planned with foreknowledge of the requirements of the situation. During the prospecting and initial development stages of a new mine, small two- or at most three-compartment shafts should be constructed. The selection of a new shaft site and the planning of suitable facilities can follow when the development has proved the possibilities for a large production. In coal mines or in mines where boring can be resorted to to secure preliminary information, shafts can be planned for the given conditions.

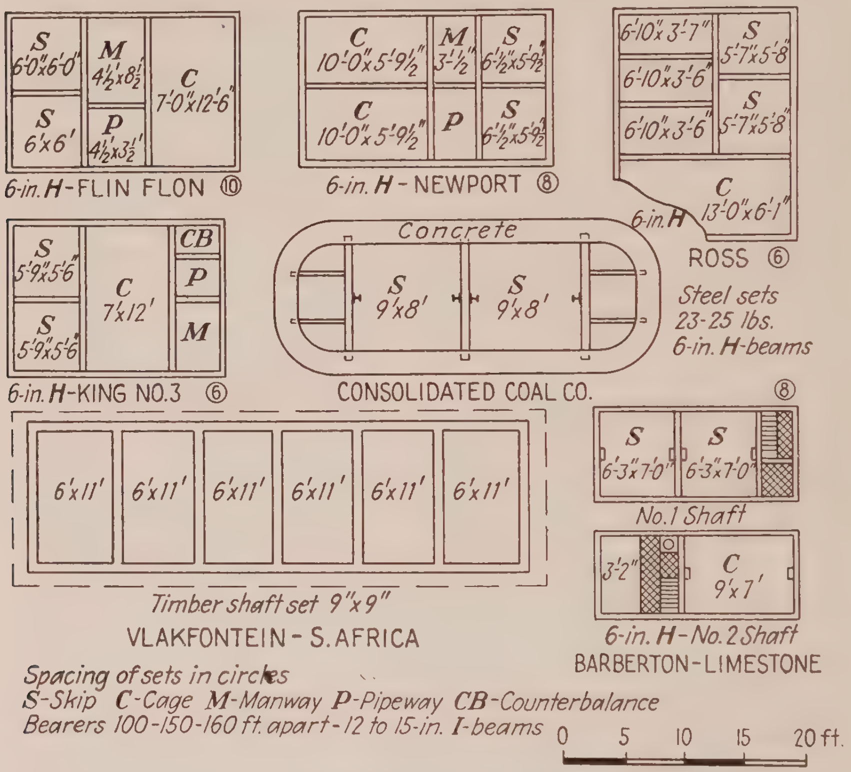

Fig. 244.—Cross sections of rectangular and elliptical shafts (upper) and circular shafts (below); the "square section" is exemplified by the Flin Flon, Newport, and King No. 3 shafts.

For mines of small output and with workings within 200 to 250 ft. of the surface, a two-compartment shaft, 5.5 to 6 ft. by 9 ft. in the clear, is satisfactory. For deeper workings a three-compartment shaft is necessary. Rectangular shafts are also shown in Chap. X.

Adits and Drain Tunnels.—Mine adits are represented by the following dimensions in the clear: 6 by 7, 8 by 8, 9 by 9, and 12 by 12 ft.. The same dimensions are satisfactory for drain tunnels. The selection of the dimensions is determined by the tonnage requirements (output), the length, the quantity of water to be handled, and the nature of the ground.

Adits and drain tunnels are constructed for single or double track. Where ground breaks well and is self-supporting, the section is made 8 by 8 ft. for an output of 500 tons or more and 6 by 7 ft. for smaller outputs. The section is more or less arched. It should be borne in mind that too small a section in hard rock is more difficult to construct than a larger one. In heavy ground which requires timbering, the section is trapezoidal and made as small as possible consistent with the tonnage to be handled. Long adits and adits that have to handle large quantities of water are made of larger dimensions than for the opposite conditions.

Drifts and Crosscuts in Metal Mines.—A single-track drift is commonly used. The smallest mine car is 2 ft. wide and clears 4 ft. above the rail. The smallest mine drift is 3.5 by 6.5 ft. in the clear. The usual dimensions are 4.5 by 7 to 5 by 7 ft. Trapezoidal sections are made 3.5 to 4 ft. at the top, 4.5 to 5 at the bottom, and from 6.5 to 7 ft. high. Where a trolley wire is used the least height should be 7 ft., and 7.5 is preferable. Crosscuts are of the same dimensions as drifts except in cases where they connect with shafts, where they may be made wide enough to accommodate a double track. In Minnesota iron mines, main haulage levels are 8 by 8 ft. to 8 by 9 in the clear and 10 by 10.5 outside of the timbers; sublevels are 7 by 8 ft. and 6 by 7; connecting crosscuts on sublevels, 2.5 by 6 ft. to 3 by 6.

Raises and Winzes.—As these are supplementary workings, the dimensions are largely controlled by local conditions, and practice affords wide variation. The raise is constructed of minimum dimensions, 4 by 4 to 4 by 6 ft. inside the timbers. In square-set mining it is usually two sets wide. In Minnesota iron mines raises are 4 by 6 ft. to 5 by 8. At the Alaska-Treadwell a 6 by 7 raise was used. Winze dimensions depend on the use to which they are to be put. Where hoisting is done, two compartments are constructed whose dimensions range from 5 by 8 ft. to 6 by 12 in the clear. For prospecting, 4 by 4 ft. to 4 by 6 may be used. Where both chutes and manways are to be accommodated in a long raise or winze, the raise or winze is made somewhat larger than the minimum dimensions given. In a given mine the most convenient dimensions for workings of this nature are quickly discovered.

Entries in Coal Mines.—Main entries range from 10 to 14 ft. wide and from 6 to 8 ft. high in the clear. The trapezoidal section is common where timbering is required. For side entries the range is 8 to 10 ft. wide by 6 to 8 ft. high. In longwall mining side entries close down until they are only 4.5 to 5 ft. high. Low cars are needed in such conditions. Between main entries a pillar from 40 to 50 ft. thick is left; between side entries, one from 15 to 50 ft.

Design of Haulageways.—The type of car, its width and height, and the least side clearances allowable, together with the necessary top clearance, determine the least width and height of the passage. At curves the width should be increased. The timbers required for support can be drafted in on the least-dimensional drawing, and the minimum outside dimensions determined.

Grades, Curves, and Turnouts.—Drifts, crosscuts, and entries are graded in favor of the loads. The grade ranges from 0.5 to 1 per cent. This is sufficient for drainage. In some metal mines crosscuts and drifts meet at right angles, and turning of the cars is provided for by a greased plate, or preferably a turntable, at the intersection. Where mechanical haulage is used curves ranging from 15 to 50 ft. in radius must be constructed. In coal mines, entries are turned off at an angle of from 45 to 60 deg. and widened sufficiently to give a smooth curve to the track at both tangents. Partings or turnouts are provided by increasing the width of the drift or entry for a length sufficient to accomodate a train of cars. A double track is placed in this length.

DEVELOPMENT RATIOS

Three development ratios can be figured and the resulting ratios used to compare different methods of development. It is evident that if the volume of the development workings and the volume of ore opened up ready for mining can be determined, the approximate cost of development per unit of ore mined can be figured. The three ratios suggested are

Table 109.—"Productive Development" Ratios for Veins of Varying Widths

Thickness of vein, ft.	1 cu. ft. development develops					
	Vertical		60° dip		45° dip	
	Cu. ft.	Tons	Cu. ft.	Tons	Cu. ft.	Tons
1	3	0.23	3.54	0.27	4.35	0.33
2	6	0.46	7.08	0.54	8.70	0.66
3	9	0.69	10.62	0.81	13.05	0.99
4	12	0.92	14.16	1.08	17.40	1.32
5	15	1.15	17.70	1.35	21.75	1.65
6	18	1.38	21.24	1.62	26.10	1.98
8	24	1.84	28.32	2.16	34.80	2.64
10	30	2.30	35.40	2.70	43.50	3.30
12	36	2.76	40.48	3.24	52.20	3.96
14	42	3.22	49.56	3.78	60.90	4.62
16	48	3.68	56.64	4.32	69.60	5.28
20	60	4.60	70.80	5.40	97.00	6.60

$$\frac{\text{Whole volume of ore developed (cu. ft.)}}{\text{Whole volume of dead-work development (cu. ft.)}} = \text{Volume of ore developed per unit of dead work}$$

$$\frac{\text{Whole volume of ore developed (cu. ft.)}}{\text{Whole volume of productive development (cu. ft.)}} = \text{Volume of ore developed per unit of productive development}$$

$$\frac{\text{Whole volume of ore developed (cu. ft.)}}{\text{Combined volume of dead work and productive development (cu. ft.)}} = \text{Volume of ore per unit of development}$$

It is, of course, not always practicable to determine these ratios, but where sufficient information is at hand they should be calculated for proposed methods of development. The productive development ratios have been approximately figured for narrow veins and are given in Table 109. The unit taken is a block 200 ft. in length and the level spacing is 100 ft. vertically. The development per block is taken as 6,500 cu. ft. The volume of crosscuts in the wider veins has been omitted. Tons have been figured on a basis of 13 cu. ft. per ton.

The following ratios for productive development have been calculated.

For top slicing (Mesabi practice), 11–12 cu. ft. per cu. ft.
For top slicing and ore caving (Newport mine), 5–6 cu. ft. per cu. ft.
For block-caving (Tobin mine), 8–9 cu. ft. per cu. ft.
Block-caving (Ohio mine), 4 cu. ft. ore per cu. ft. of block development
For panel working, room and pillar (6-ft. coal seam), 10 cu. ft. per cu. ft.

Various methods of representing the relation between development and ore mined are employed in mining practice. From technical journals and other sources the following have been gathered:

Pittsburgh South Peak (Nevada).	1 ton waste to 4.77 tons ore 1 ton ore by development to 10.5 tons ore mined
Round Mountain (Nevada)	1 ft. for each 7.35 tons milled
Mammoth mine (California)	1 ton ore to 1.45 tons waste 1 ft. development to 2.6 tons ore mined
Alaska-Treadwell (Alaska)	1 ft. development to 105 tons ore mined
Alaska United Gold Mining Co. (Alaska)	1 ft. development to 111 tons ore mined
Alaska Mexico (Alaska)	1 ft. development to 55.6 tons ore mined
Average of three	1 ft. development to 98 tons ore mined (1 cu. ft. to 1.46 tons) (1 cu. ft. to 19 cu. ft.)
Copper Range Co., 1911	1 ft. development to 66.2 tons ore milled
North Star mine (California)	1 ton waste to 10.3 tons mined
Harold mine (Minnesota)	1 ft. development to 63 tons ore 1 cu. ft. development to 8.2 cu. ft. ore mined

SHAFT SINKING

Classification of Material Penetrated and Water Conditions.—The methods used in shaft sinking are selected with special reference to the material that must be excavated and supported. The presence of moderate quantities of water calls only for the addition of bailing or pumping facilities to the sinking equipment, but where excessive quantities are encountered a radical change in method is often necessitated. The following classification of conditions is used.

a. Hard rock—little or no water
b. Hard rock—more or less fissured—and with moderate amounts of water
c. Hard rock—more or less fissured—and with excessive amounts of water
d. Soft nonplastic rocks, sandstones, limestones
e. Soft nonplastic rocks, sandstones, limestones—fissured and with moderate amounts of water
f. Soft nonplastic rocks, sandstones, limestones—fissured and with excessive quantities of water
g. Soft semiplastic rocks, clay shales, shales
h. Noncoherent material, sand—without water or with only moderate quantities
i. Noncoherent material with excessive quantities of water; quicksand, mud, ooze, and material of similar nature

Methods of Shaft Sinking.—The methods of shaft sinking are:

1. Drilling by hand; blasting
2. Drilling by machine drill and blasting
3. Raising
4. Forepoling; spiling; sheet piling
5. Forepoling and bottom boarding
6. Steel sheet piling
7. Boring methods
8. Drop shaft or open caisson, with hand excavation, orange-peel bucket, or sack borer
9. Compressed-air caisson
10. Freezing
11. Cementation

Selection of Method for Typical Conditions.—Shaft-sinking methods have been evolved from practice and experience. Unusual conditions necessitate the modification of ordinary methods or the invention of new ones. The methods characteristic of modern mining practice offer some kind of solution for almost any combination of conditions. In the tabulation which follows an attempt is made to suggest the most likely method for the conditions described before.

a. Methods 1, 2, 3

b. Methods 1, 2, 3, with use of sinking pumps or bailing tanks, or both in methods 1 and 2; method 3 would provide for drainage; close timbering

c. Methods 1 and 2 and heavy pumping equipment; or methods 1 and 2 in conjunction with method 11. Close timbering would be required. Advance dewatering by bores close to shaft area may be very effective

d. Methods 1, 2, 3

e. Methods 1 and 2; sinking pumps or bailing tanks; close timbering

f. Methods 1 and 2, with heavy pumping equipment; or methods 1 and 2 in conjunction with method 11

g. Methods 1 and 2, with heavy timber or steel and concrete supports

h. Methods 4, 5, 6

i. Methods 8, 9, 10, 11 in some cases

Description of Methods of Shaft Sinking.—1. *Hand Drilling; Blasting.*—The bottom of the shaft is carried in a series of benches, the first bench or sump being driven in advance across the width of the shaft. The drill holes range from 3 to 4 ft. in depth. The work follows in orderly sequence—drilling, blasting, mucking, and timbering. In soft materials the pick, shovel, and bar are used for excavation.

2. *Machine Drilling and Blasting.*—This is the usual method employed for medium hard to hard rocks. The sequence of operations is as follows: Lowering lights (where electricity is used), drilling, setting blasting boards on timbers, removing men, charging holes, firing, removal of powder smoke, and mucking. Timbering follows the excavation at distances varying from 10 to 50 ft. With fissured rock the timbering follows close down upon the excavation, which is kept as close to the outer line of the lagging as possible. Holes are drilled from 4 to 8 ft. in depth. Hand-held automatically rotated drills are used, receiving air from a manifold on the air pipe. From 2 to 12 drills may be used, the number depending upon the cross section of the shaft and the rate of sinking desired.

Improvements in the usual method of handling the broken rock by means of the bucket and shoveling have been made in recent practice. H. R. Drullard describes the following method, used in Butte. An extension is used upon a sinking cage, enabling it to be lowered to the bottom of the shaft. Extension runners up to a length of some 30 ft. are attached to the cage, giving a corresponding range below the ends of the guide timbers. A mine car is used on the cage. The side of the cage is redesigned so as to give a free opening at the side at the top edge of the car body. An auxiliary panlike bucket, 36 in. in length, 29 in. in width, and 18 in. in height is used on the shaft bottom. The broken rock is shoveled into this and hoisted by a small electric or air hoist, which is placed two or three sets above the lowest timber set and controlled by ropes attached to the throttle and brake. The discharge end of the bucket is attached by links or short chains to the side of the cage, and as the bucket is raised it tips up and discharges into the car. A considerable saving in time and effort results by the use of this rig, which is shown in Fig. 245. A skip is also used in a similar manner. Hubbell describes the use of a short steel slide and a scraper, the combination being used for loading a sinking bucket.

3. *Raising Shafts.*—This method can be applied only where underground workings can be extended to the shaft section. A timbered raise is driven. Shaft timbers are used, and the timbering of the shaft is completed as the raise is driven upward. A working platform is placed upon the uppermost set; one compartment is boarded up on the sides and used as a waste chute and the other as a manway. Air pipes are extended upward for ventilation. A type of round is used for blast-

ing similar to that required in sinking. The advantage of the system is that the handling of the waste is much simplified and is done at a minimum cost. Greater speed can also be attained. Stoping drills are used. The sequence of steps is as follows: Removal of spoil from working platform; setting of timbers; drilling with stoping drill; placing blasting boards on timbers; removing drills and air hose; charging holes; firing; removal of gaseous products of blasting; placing working platform

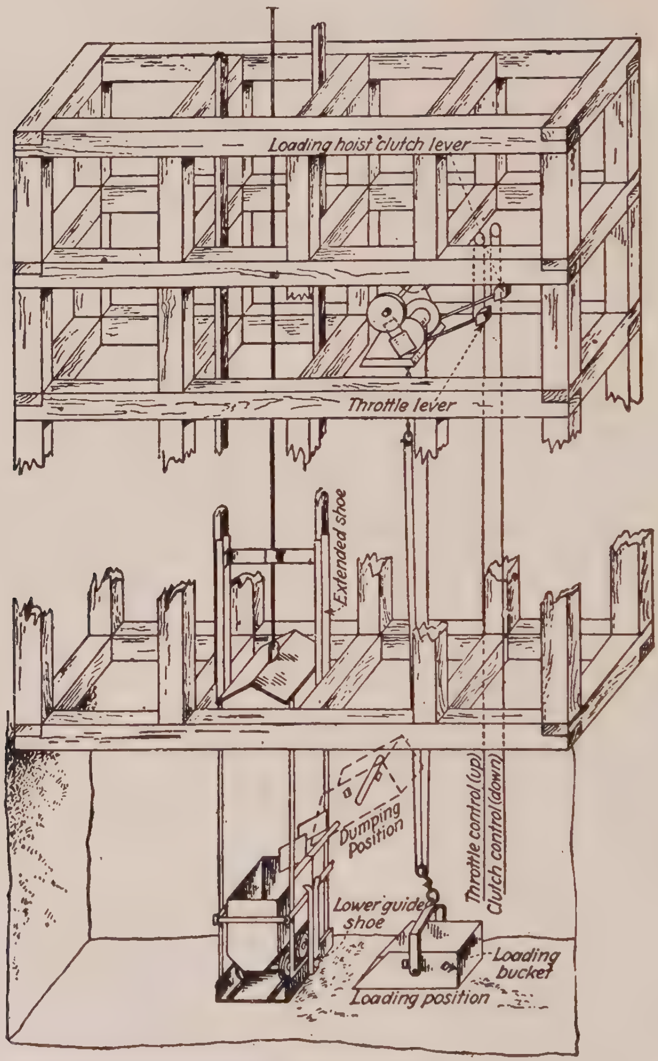

Fig. 245.—Butte method of shaft mucking. (*H. R. Drullard.*)

upon timbers. The method is impracticable in soft or loose ground. In large shafts, part of the shaft is sometimes sunk and the remainder stripped down.

4. *Forepoling.*—In starting a shaft in soft ground, the sides of the excavation can be protected by driving a wall of sheet piling about the shaft. The enclosed earth is removed, the timber sets are placed in position, and the sheet-piling wall is braced. Within the first line of sheet piling another line is driven and excavation and timbering proceed. In the sinking of a deep shaft, spiling may be driven in advance of the excavation, to support the excavation temporarily while the main shaft sets are being placed in position. Spiling consists of 2- by 6- or 3- by 6-in. boards, 6 ft. long. One end is chisel-shaped.

A modification of this method consists in using solid cribbing, 12- by 12-in. timbers bolted together and suspended from supports above. At the bottom a steel sleeve similar in principle to the shield used in tunnel driving is used. This is constructed of ½-in. steel plates and is provided with a cutting edge and a heavy timber or steel bracket. Jack screws are used between the lowest crib set and the bracket, and the steel sleeve is forced down. As soon as a sufficient advance has been made another crib set is placed. Figure 246 shows the principle of the arrangement. Where the sinking is in sand containing much water, a steel drum 24 to 36 in. in diameter is forced down in the bottom and used as a sump in order to drain the sand in advance of the working.

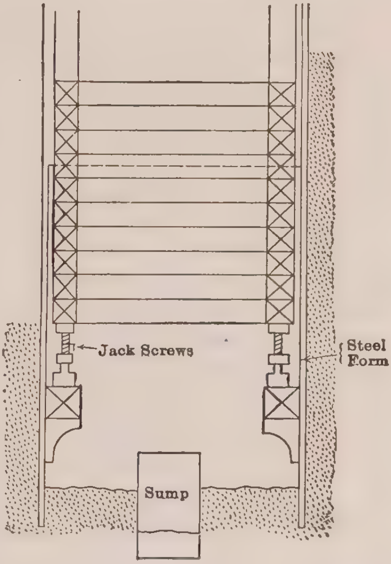

Fig. 246.—Shaft sinking in difficult ground.

5. *Forepoling.*—In quicksand, bottom boards are used in order to prevent the sand from being forced in. They are used in conjunction with spiling and are braced by struts extending back to the timbers already in place. One bottom board is advanced at a time until a sufficient advance has been made for a new set.

6. *Steel Sheet Piling.*—United States Steel, Lackawanna, or other forms of steel sheet piling can be substituted for wooden sheet piles. By their use shafts can be sunk more satisfactorily than with the wooden sheet piling. The extreme length of steel sheet piling ranges from 60 to

80 ft. These piles are driven with a pile driver to their full depth before the material enclosed is removed. As the shaft is excavated the wall of sheet piling is braced by wall plates and dividers suspended from bearers at the surface.

7. *Boring Methods.*—The Kind-Chaudron method of boring a shaft was in effect boring by a large-diameter churn drill of great weight. It possessed the advantage of sinking and lining without dewatering but required highly specialized equipment. It is now of historic interest only. See "Shaft Sinking under Difficult Conditions," by Riemer, Corning, and Peele, and "Modern Mining Practice," by Bailes-Cousins, vol. 3, p. 129.

The J. B. Newsom method of shaft boring is based on chilled-shot boring, described in a previous chapter. Newsom started work in 1934. His method is applicable to hard, firm rocks which are not water-bearing or which can be drained. Softer rocks that do not swell and some water flow may be handled. The maximum diameter of bore attempted is 5.5 ft.; depths attained were 1 152 ft. at Grass Valley and 1,208 ft. at the Zenith mine.

The cutting tool (core barrel) is cylindrical, 15 ft. in length; the cutting ring, 1½ in. in thickness and riveted to the shell. Notches are cut in the shoe to afford openings for the entrance of the chilled shot, which is fed down into the kerf. The relatively short drilling bar that rotates the core barrel is driven by a turntable, a gear train, and a 100-hp. motor contained in a cylindrical cage, which is attached to bails and a hoisting rope. Jackscrews pressing against the wall of the bore prevent turning of the cage. The speed of the cutting tool is 52 r.p.m., weight 6 tons, and length inside the core barrel with new shoe 15 ft. 2 in. Power is supplied by cable. An air hose provides air for the operator. A man cage, muck bucket, bailer, core puller, essential safety appliances, hoist, derrick, and reels for power cable and air hose complete the equipment.

When the bore reaches the limit imposed by the length of the core barrel or by the presence of slip planes, boring is stopped, the jacks are loosened, and the cage and core barrel are hoisted to the surface; cuttings are bailed out and the core is broken by wedging. A corer is lowered over the core and it is hoisted to the surface. Two operators are required per shift, one in the drilling cage and the other at the surface. Experience obtained in three bores has greatly improved drilling equipment and technique.

The Idaho-Maryland bore required 21 months for its completion to 1,152 ft. The Zenith ventilating shaft required less than 7 months for 1,200 ft., 11,044 man-hours in all, making an average advance of 5.8 ft. per day. Direct cost was $19.50 per foot or 82 cents per cubic foot; labor and supervision was $12.16 and power and materials $7.34 per foot.

Chilled shot consumed was 32 lb. per foot. Many other details are given in *Technical Publication* 1068, *A.I.M.E.*

8. *Drop Shaft.*—The caisson consists of a hollow cylinder constructed of steel, timber, brick, or concrete. The lower edge is a steel-cutting shoe. The shoe is placed in position and the first section of the caisson is constructed. It is then sunk by excavating the material within by hand or by dredging with an orange-peel bucket. The weight of the caisson causes it to settle and follow down with the excavation. As the caisson sinks additional sections are constructed. The frictional resistance opposing the movement of the caisson has been determined in a number of cases. The three factors are the nature of the exterior surface of the caisson, the material sunk through, and the depth. Sand offers great resistance, muds and clays less.

Frictional Resistance per Square Foot of Surface[a]

Cast-iron cylinder in gravel	200 lb.
Cast-iron cylinder in sand and river mud	400 lb., 600 lb. small depth
Cast-iron cylinder in sand and gravel	800 lb., 1,000 lb. for 20 to 30 ft.
Pneumatic caisson in silt, sand, and mud (40 to 80 ft. depth)	280 lb., 350 lb.
Pneumatic caisson of Brooklyn Bridge	600 lb.
Five shafts sunk on the Catskill Aqueduct	300 to 400 lb.
	630 lb. for 95-ft. depth
	751 lb. for 116-ft. depth
	1,411 to 872 lb. for depth of 49 to 79 ft.

[a] *Eng. Contr.*, vol. 39, p. 391.

In some cases it may be necessary to load the caisson to the extent of 2,500 to 3,000 lb. per square foot of rubbing surface.

9. *Compressed-air Caisson.*—The method differs from the preceding in that a closed chamber, the working chamber, is placed on the bottom of the caisson. This chamber is filled with air under sufficient pressure to prevent the entrance of the water. Access to the working chamber is obtained through air locks, by which men enter and material is removed. Air pressure must be maintained constantly during sinking operations and the men must work in the compressed-air chamber. Only robust men can endure such conditions. The method is limited to depths not exceeding 125 ft., or when the air pressure required does not exceed 45 lb. per sq. in.

10. *Freezing Method.*—This is known as the Poetsch method (invented in 1883). It is applied to soft unstable ground carrying large quantities of water. Boreholes are sunk on a circle of somewhat larger diameter than the shaft and at intervals of 31 to 39 in. apart. Within these boreholes pipes for the circulation of the brine are installed. Two pipes are necessary, the inner one 1 in. in diameter, the outer 4 to 5 in. in diameter. Two circular manifolds connect the outer and inner pipes,

and branches connect with the refrigerating system. Usually an ammonia refrigerating plant is used. The brine which is cooled at the refrigerating plant is either calcium or magnesium chloride solution. The size of the plant varies. One plant, capable of producing 60 tons of ice per day, consisted of four 40-ton units with a combined freezing capacity equal to 160 tons of ice. Freezing requires from 2 to 4 months, depending on the size of the shaft. S. F. Walker summarized the principal facts concerning the absorption of heat as follows:

> Assuming that the ground is saturated with water:
> Volume of water, 37 per cent or 19.6 per cent by weight.
> Volume of sand, 63 per cent or 80.4 per cent by weight.
> Specific heat of water, 1.0.
> Specific heat of sand, 0.2.
> Specific heat of ice, 0.5.

Approximately 2000 B.t.u. will be required to absorb the heat from 1 gal. of water and change it into ice. By assuming that an ice cylinder is equal to twice the diameter of the shaft in the clear and of a depth equal to the depth of the freezing tubes plus two-thirds the clear diameter of the shaft, the approximate weights of sand and water to be frozen can be calculated, and from these weights the total heat to be absorbed can be determined. More or less heat will be lost by conduction and by the circulation of the water. Once the ice cylinder has been formed the refrigeration is kept up, but at a lower rate. It is necessary to operate the refrigerating plant throughout the period of sinking and tubbing. Walker estimates that in one case 1,600,000,000 B.t.u. was absorbed in freezing an ice cylinder (17,000,000 B.t.u. per day). The progress of freezing can be determined by measuring the rate of circulation of the brine and measuring the temperatures of the ingoing and outgoing brine.

The depth to which it is possible to go is given by various authorities as ranging from 787 to 1,300 ft. Solid ice at 20°C. and under a pressure of 20 atmospheres (656 ft.) becomes plastic like clay. Mixed with sand and clay, its plasticity is less. The difficulties arising are the deflection of the bores, the breakages of the pipes due to contraction, and the presence of saline waters. This method is used where others fail.

In sinking in the frozen cylinder, light powder charges are used and a temporary wooden timber support is carried down. The permanent cast-iron lining is started from the bottom of the shaft and carried up, the temporary wooden lining being removed.

Oetling Freezing Method.—This method involves the idea of freezing the ground in sections as the sinking proceeds. The permanent lining is put in as the shaft is sunk. The freezing apparatus is a cylinder equal in diameter to the shaft and 44 in. in height, with the lower end closed by a plate. The cylinder is in sections, each of which can be removed. Each section is provided with freezing coils. After freezing the ground, two sections are removed, the ground is thawed locally and removed, and a segment of the lining is inserted. This process is repeated.

11. *Cementation.*—Cement injection of the rock zone surrounding a proposed shaft site as a preliminary to actual excavation, the purpose being to seal off water, is a comparatively recent method. The success that has attended its use gives it prominence as an important adjunct of shaft sinking. In principle it is nothing more than grouting on a large scale. The method is applied to shaft sinking in the following manner. Six or eight boreholes are driven in a circle surrounding the shaft section. The circle is from 2 to 4 ft. greater in diameter than the inside diameter of the shaft. The upper portion of each borehole is lined with a heavy steel casing that is provided with a cast-iron head containing an inlet connection for the cement injection pipe and an outlet pipe and valve (Fig. 247). A churn drill is used for sinking in hard or brittle rocks; augers are used in soft rock. The cementing is done in stages. After the hole has been deepened the cement injection pipe is lowered almost to the bottom, the cast-iron head is attached, and a hose connection is made between the pipe and the pump. Clear water is then pumped through the injection pipe and discharged at the outlet on the casing head. When the bore is thoroughly clear, the outlet on the casing head is closed and the pump is worked to its full capacity in order to clear the crevices and fissures as much as possible. About half an hour is required for this. Then, without disturbing the connections, a mixture of cement and water (5 per cent by weight) is forced through the pipe. After about an hour the proportion of cement is increased to 10 per cent. The filling of the fissures with cement is indicated by a rise in the pressure shown on the gage attached to the casing head or the pump. When the pressure rapidly rises, indicating the almost complete filling of the fissures, the outlet on the casing head is gradually opened and the excess cement is returned. Cementing is completed when the outlet valve must be fully opened in order to keep the pump in operation. Clear water is then pumped through the injection pipe, and the pump, pipe, and bore are

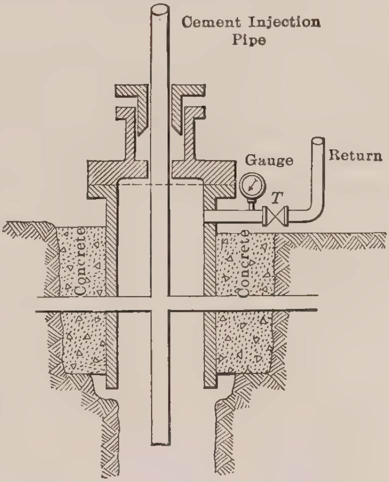

Fig. 247.—Pipe arrangements for cementing shafts.

cleared of the excess of cement. The injection pipe is removed and the bore is left for 8 to 10 hr., or until the cement sets. The surplus cement mixture returned is used on another hole. Where the pressure suddenly drops during cementation, indicating the breaking through of cement in a fissure or the opening of a channel to the surface, a mixture of 30 per cent is pumped under moderate pressure until the pressure begins to rise. The length of the advance of the borehole that can be cemented in a single stage depends primarily upon the stability of the walls of the bore. This advance is 3 to 20 ft. or more. If the bore stands well, it may be sunk to the required depth and cemented at one operation. The result of the cementing is to surround the shaft excavation with a wall of grouted rock, within which the shaft can be sunk by ordinary methods. There is a close resemblance to the freezing method, in that the frozen water binds the rock mass together in much the same manner as the cement in setting binds the aggregate in concrete. The pump used is a duplex plunger pump provided with steel ball valves, as ordinary rubber valves wear too rapidly to be used. The quantity of cement required varies with the volume of the fissures to be filled; a badly broken rock mass would of course require a correspondingly greater volume of cement. A few large fissures may be of considerable extent and must be filled. The following table illustrates the quantity required for a single borehole in one case.

TABLE 110

Depth		Rock	Cement absorbed, lb.	Cement returned, lb.	Pressure per sq. in., lb.
Ft.	In.				
16	0	Earth			
32	0	Crumbly chalk	27,700	700	71.0
49	0	Crumbly chalk	46,300	2,000	14.2–142
65	0	Crumbly chalk	48,200	3,000	42.6–183
81	0	Chalk and flints	42,900	3,000	48.4–128
97	6	Chalk and flints	33,200	4,400	42.6–170
114	0	Chalk and flints	23,600	4,000	42.6–7.1
130	0	Chalk and flints	33,900	6,000	42.6–185
146	0	Chalk and flints	7,400	....	85.2–121
260	0	Gray fuller's earth			
		Total	263,200	23,100	

Where water-bearing fissured zones are penetrated in shaft sinking, the removal of the water can sometimes be obviated by drilling diamond-drill holes in the shaft bottom and cementing the fissures in a manner similar to that described. The method has the decided advantage of requiring a small and simple plant involving a relatively small investment as compared with the plant required for the freezing method. Hard, brittle, or granular rocks can be cemented in the manner described. Clayey rocks, marls, and plastic rocks generally offer difficulties.

Relative Importance of Sinking Methods.—In self-sustaining rocks, hand-held drills and sinkers are standard for drilling. Blastholes and and blasting are planned to cut down overbreak and wall weakening as much as possible, and steel blasting shields are generally used. Mucking is by shovel into loading bucket or tray (Drullard method, p. 521), which is hoisted and discharged into a skip or car on a cage with extension shoes long enough to reach the shaft bottom, or by the older method of shoveling directly into the shaft buckets. J. M. Riddell and G. A. Morrison recently described a method of mechanical mucking (*Technical Publication* 1778, *A.I.M.E.*) in which a small clamshell bucket, 7.65 cu. ft., is operated by a hoist supported upon a movable platform, which in turn is on a frame attached to the lowest shaft set; the operator can move the bucket along the long axis of the shaft and thus serve most of its area. The loaded clamshell is discharged into a 32-cu. ft. round shaft bucket. The rig was tried out on 1,126 ft. of 8- by 17-ft. shaft section. Mucking rate per hour per man in cubic feet was as follows: Hand method 37.2 and 29.1, mechanized 211.7 and 152.2, for shale and limestone, respectively.

Difficult shaft sinking in surface alluvials is usually handled by steel sheet piling for support in advance of excavation by clamshell bucket, by drop shaft or open caisson, or by some modification of the movable open-bottom shield (p. 523). Where much water and bad ground extend beyond the limits of these methods freezing a deep cylinder of ground and excavating the shaft therein is in order although both expense and difficulty are considerable. Deep sinking in fractured water-bearing formations where the quantity of water precludes pumping is by advance cementation in stages through long drill holes followed by ordinary methods of shaft excavation.

The Newson method of boring is advantageous for ventilating outlets to the surface and exits for safety purposes. Underground chilled-shot holes have been used for ventilating connections between levels, but the greatest diameter used is some 4 ft. and the depth between 200 300 ft. At Butte a 36-in. bore connected with bottom levels and facilitated ventilation; at the United Verde mine a 48-in. bore is used for the same purpose. In both mines the calyx drill was used. J. B. Pullen gives the cost at United Verde for a 48-in. calyx borehole as $10.10 per foot or 80 cents per cubic foot.

Probably the earliest use of a churn drill underground was described by G. N. Bjorge in 1914. It was used for ventilation connections and prospecting and later for advance draining of levels in heavy water zones.

Rate of Sinking Shafts.—The rate of shaft sinking varies between wide limits. Ground conditions, depth, method, and local conditions influence the rate. Usually when work is started it is continuously prosecuted and two or three shifts per day are the rule. Shafts are sunk in heavy ground or in very wet ground at a much lower rate than where the ground is firm and free from water. Special methods of shaft sinking for difficult conditions are slow and the rate of sinking exceedingly low. Under normal conditions with dry shafts a rate of 70 to 100 ft. per month can be obtained. With a water inflow of 150 gal. per min. the rate falls to 50 ft.; where it is 500 gal., to 25 ft. per month. On the New York aqueduct a shaft was sunk in granite 183 ft. in 27 working days. At the New Kleinfontein shaft on the Rand, a maximum rate of 213.5 ft. was attained in 1 month.[1] The comparatively shallow shafts (250 to 350 ft.) in Minnesota iron mines are sunk at the rate of 40 ft. per month.[2] In soft-ground sinking with much water a rate of 10 ft. per month is considered good progress. By the Kind-Chaudron system of shaft boring the rate of sinking ranges from 10 to 30 ft. per month. Caisson sinking ranges from 10 to 50 ft. per month.

The maximum shaft-sinking rate was obtained in the Water Lily shaft of the Chief Consolidated Mining Co. near Eureka, Utah. The record was 427.5 ft. in 31 working days. The shaft was dry and the excavation was made in porphyry of average hardness.[3]

Details of Raising.—The example is taken from an article describing the Treasury tunnel raise, Ouray County, Colo. The raise was driven from an adit, 5,500 ft. long, on a vein to a vertical distance of 853 ft. above the adit level. The dip of the vein ranged from 45 to 60 deg. The dimensions of the raise were 11 by 5 ft. It was divided into two compartments, one 5 by 5 ft. for a rock chute and one 4 by 5 ft. for ladder, bucket slide, and compressed-air pipe. A 12-in. stull divided the chute from the manway. An 8-in. stull was placed on the other side of the manway. The stulls were placed in hitches at 5-ft. centers. Three-inch plank lagging divided the manway from the rock chute. A heavy floor (6- to 10-in. square timbers) placed on the topmost stulls served to protect the manway during blasting. The rock chute was kept almost full; only enough material was drawn out to take the rock broken from a round. A small compressed-air hoist served to raise timbers and tools, which were handled in a bucket. Two miners and two timbermen were the working crew on each of two 8-hr. shifts. Drilling was done with two 3C Waugh stopers. The round consisted of eight to nine 5½-ft. holes and required 2.5 hr. to complete. The cut holes were placed above the rock chute, and the other holes were so placed as to throw the rock into the rock chute and away from the manway. Several small chambers were constructed at different points on the manway and used during blasting.

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[1] *Eng. Min. Jour.*, July 29, 1911, p. 223.

[2] *Bull.* 1, *Minnesota School of Mines*, p. 73.

[3] *Eng. Min. Jour.*, vol. 113, p. 61.

Shaft sinking by raising is accomplished by raising over the full section, or by driving a small raise and then stripping down the remainder to the section using the raise as a rock chute. In stripping down, vertical holes are drilled on the bench with hand-held rotating hammer drills, or air-feed stopers can be used in the raise to drill holes at a 45-deg. angle through the sides of the raise and into the bench. In the latter case a platform has to be constructed in the raise for each round.

Cost of Shaft Sinking.—The cost of shaft sinking is commonly given in terms of cost per foot. The unit cost cannot be conveniently compared without specific knowledge of the cross section. For this reason the unit of cost based upon the cubic foot has been introduced in this book and used as far as practicable. The area of the shaft section is taken as the area of the timber set measured on the outside, plus 20 per cent of this area. Specific data on the sectional area of the excavation are used instead when they are obtainable.

Unit cost is influenced by the depth of the shaft, the area of section, the hardness of the ground, the support required, the amount of water to be handled, the efficiency of the workers, the wages paid, and the cost of materials required. Unit cost steadily increases with the depth. The larger the shaft section, other things being equal, the smaller the cost per cubic foot. The cubic foot cost is greater for shafts of small cross section than for shafts of normal section (say 130 to 150 sq. ft.). It decreases very slightly as the area is increased above the normal area or, *i.e.*, it approaches a constant value. The hardness of the ground is not a very important factor. In very soft ground the extra expense for support overbalances the saving in excavation, while in very hard ground the decrease in the expense for support makes up for the increased cost of breaking. Hard ground requires increased capital for compressors and drills that are unnecessary in soft ground. The support required is an important factor. Heavier timbers and a greater proportion of labor are required in heavy ground. The rate of sinking is also lower and the overhead expense per unit is consequently greater. Where the amount of water to be handled exceeds a nominal quantity, the unit cost is rapidly increased. The cost of pumping and the inevitable delays occasioned in all the operations by the handling of the pumps and pipes, as well as the increased wage demanded by miners for wet shaft work, are the factors that make the increased cost. Where the quantity of water reaches 4,000 or 5,000 gal. per min. the unit cost may quickly reach a prohibitive figure. Efficiency is secured by good organization and superintendence and the use of a bonus system of payment. Increased efficiency operates to decrease

TABLE 111.—COSTS OF SHAFT SINKING

No.	Mine	Depth, ft.	Description	Cost per ft.	Cost per cu. ft.	Rock material	Conditions	Reference
1	Daisy shaft, Goldfield, Nev.	100	4 × 8 in clear	$32.45	$0.52	Hard andesite	High labor and materials cost; timbered	*Eng. Min. Jour.*, vol. 84, p. 1106
2	Edgerton, Wis.	153	5.5 × 9 inside 2 compts.	19.06	0.32	Drift and limestone	63 ft. sunk by machine; 90 ft. by hand, same conditions as above; 40 ft. timbered	*Eng. Min. Jour.* vol. 81, p. 1234
3	Ellison shaft, Homestake mine, S. D.	....	10 × 18 inside 3 compts.	60.64	0.21	Hard rock	Timbered 12 × 12 timbers; machine drilling	*Min. Sci. Press*, vol. 88, p. 127
4	Argonaut shaft, Calif.	283	3 compts. 4–1 × 5–9	43.11	0.24	Diabase, schist, slate	Timbered 20 × 20-in. wall-plates; incline 60°	*Eng. Min. Jour.*, vol. 91, p. 514
5	Hibbing dist., Minn.	213	2C 5 × 6 1C 6–4 × 6	54.60	0.25	Glacial drift; taconite	Timbered	*Bull.* 1, *Minn. School of Mines*
6	Chisholm dist., Minn.	253	2C 5 × 6 1C 6–4 × 6	82.60	0.375	Glacial drift; taconite	Timbered	*Bull.* 1, *Minn. School of Mines*
7	Woodbridge shaft, Buhl, Minn.	100	29-ft. outside diameter	500.00	0.75	Sand; gravel; quicksand	Compressed-air caisson	*Bull.* 1, *Minn. School of Mines*
8	Rogers, Iron River, Mich.	140	27-ft. outside diameter	500.00	0.87	Shifting overburden; gravel; sand; quicksand	Concrete drop shaft—compressed air in part	*Min. Sci. Press*, vol. 103, p. 831
9	Brier Hill, Vulcan mine, Mich.	Surface to 62 62–549.5 549.5–635.34	Circular, 14-ft. inside diameter	104.01 76.28 87.19	0.351 0.257 0.294	Drift Moderately hard rock Hard rock	Concrete lining, av. thickness 18 in.; minimum thickness 6 in. in rock; actual average 19 in.; constructed by 6 × 8 raise and sinking remainder	*Trans. L.S.M.I.*, vol. 14, p. 145
10	Modderfontein, Rand, South Africa	2,285	Circular, 18 ft. inside	106.00	0.301	....	Hand and small machine drills; brick lining	*Min. Sci. Press*, vol. 106, p. 785
11	Cinderella Deep, Rand, South Africa	0–1,000 1,000–2,000 2,000–3,000 3,000–3,900	4C 5 × 6 1C 6.5 × 6	87.08 95.28 109.32 126.10	0.26 0.28 0.32 0.37	Quartzite Quartzite Quartzite Quartzite	Timbered	*Eng. Min. Jour.*, vol. 82, p. 1060
12	16A, Crown mines, South Africa	3,500	20-ft. inside diameter 9-in. concrete blocks	169.00	0.40	Hard rock	White boss, native labor	*Third Empire Min. Met. Cong., South Africa*
13	Randfontein shaft, South Africa	3,421	22-ft. inside diameter concrete lining	159.00	0.37	Hard rock	White boss, native labor	*Third Empire Min. Met. Cong., South Africa*

Table 111.—Costs of Shaft Sinking.—(*Continued*)

No.	Mine	Depth, ft.	Description	Cost per ft.	Cost per cu. ft.	Rock material	Conditions	Reference
14	McPherson shaft, Ducktown, Tenn.	57	3 compt. 8 × 17 ft.	65.14	0.34	Hard rock	Month's sinking	*Eng. Min. Jour.*, vol. 129, p. 301
15	Ajax shaft; Colo.	502.5	6 ft. 2 in. × 15 ft. 3 compt.	57.30	0.38	Granite	Timbered 1,481 to 1,983.5 depth	*Eng. Min. Jour.*, vol. 120, p. 255
16	American Eagle shaft, Victor, Colo.	600	7 × 17 outside dimensions 3 compt.	38.75	0.32	Hard rock	Normal	*Eng. Min. Jour.*, vol. 122, p. 657
17	Hallnor, Ontario	997	19.8 × 7.6—3C 2S; 1M & P	84.90	0.56	Slate and bedded arkose	Timbered; beneath pentice	*Trans. C.I.M. & M.*, vol. 44, p. 12
18	King mine No. 3, Canada	1,153	18 × 13—5C 2S; cage; 2M & P	186.43	0.77	Serpentinized peridotite	Steel shaft sets	*Trans. C.I.M. & M.*, vol. 42, p. 371
19	Ross shaft, Homestake mine, S. Dak.	3,241.5	14 × 19.25—6C 2S; cage; 3M & P	132.31	0.49	Hard rock	Steel shaft sets; pilot raise and stripping	*T.P.* 621, *A.I.M.E.*
20	Flin Flon, south main shaft	3,000	13.5 × 19.5—5C 2S; cage; 2M & P	164.58	0.62	Quality porphyry	Steel shaft sets; full section, pilot raise, pilot winze and stripping	*Trans. C.I.M. & M.*, vol. 44, p. 478
21	U.S. Mine, Bingham, Canada	410	18.5 × 6.33—3C	38.48	0.39	Hard limestone; porphyry	Timbered, internal or subshaft	*Mining and Metallurgy*, August, 1933
22	Falconbridge Ontario	125	26.25 × 19.9	282.00	0.54	Gravel; sand; boulders	Sinking shoe, concreted timber	*Trans. C.I.M. & M.*, vol. 41, p. 103
23	Magma No. 5	2,531	21 × 7.5—4C	116.44	0.74	Quartzite; shale; diabase	Timbered	*Inf. Circ.* 6168, *U.S. Bur. Mines*

Note.—C is compartment; S is skipway; M & P is manway and pipeway. Cost per cubic foot was determined by dividing the cost per foot by the area of shaft measured to outside of timbers; it is not the cost per cubic foot of excavation. Costs for the Flin Flon shaft were given as follows: Shaft $484,868.82; headframe $161,845.19; bins and structure $169,360.98; hoist house $133,882.83; hoist equipment $801,834.84; generator and fan room $39,640.79; change house and equipment $29,103.15; water supply and pipe lines $37,602.90; total cost $1,808,139.50.

direct unit costs, and by securing a high rate of sinking it indirectly decreases overhead and general charges. The wage is less important as a factor than the securing of experienced and efficient workers. Shaftmen usually receive higher wages than ordinary miners and it is on the whole an economy to pay a sufficient wage to attract the best workers. Increase in the cost of supplies results in a proportional increase in the unit cost. The powder, timber, and other supplies, assuming efficient use, are a constant quantity for any given case.

Where shafts exceed a nominal depth (500 to 1,000 ft.) a high rate of sinking should be attained with the object of reducing the general charges for plant, superintendence, and engineering.

Examples of Cost of Shaft Sinking.—Shaft-sinking costs are reported in a variety of ways. Often specific data are lacking, and as a consequence it is difficult to compare costs. A general table, Table 111, has been compiled in which a number of costs have been brought together. In most cases the costs represent direct costs and overhead; plant and engineering are not included.

The cost of raises is usually less per cubic foot than the cost for sinking, because mucking is reduced to a small factor. Examples of cost in metal mines are given in Table 112. The costs given in the table are estimated for various sections and different methods of support.

TABLE 112.—ESTIMATED COST OF RAISING[a]

	Rock, 4½ × 4½ ft. 5 × 5 ft.	Stull, 5 × 8 ft.	Small, cribbed, 4½ × 9 ft.	Square-set, 7 × 12-ft., 10 × 10-in. timber	Large cribbed, 6 × 11-ft., 5 × 8-in. timber
Direct labor:					
Drilling	$1.90	$ 2.50	$ 2.50	$ 3.25	$ 2.80
Tramming	0.40	0.65	0.60	1.35	1.05
Timbering	0.30	0.60	0.80	2.50	1.90
Indirect labor	1.75	2.25	2.25	3.85	3.20
Insurance	0.13	0.18	0.18	0.35	0.25
Explosives	1.40	1.90	1.90	2.30	2.15
Timber	0.24	1.32	2.12	6.08	8.80
Drill-steel expense	0.50	0.70	0.70	1.30	1.20
Drill repairs	0.35	0.50	0.50	0.80	0.75
Power	0.50	0.75	0.75	1.40	1.20
Miscellaneous supplies	0.60	0.75	0.75	0.95	0.95
Totals	$8.07	$12.10	$13.05	$24.13	$24.25

[a] M. J. Elsing, *Eng. Min. Jour.*, vol. 133, p. 78.

TUNNELING AND DRIFTING

The methods used in driving adits, drifts, crosscuts, entries, etc., are relatively simple. They are best described with reference to the rock materials through which the working must be extended. The rock materials are hard rock, soft rock, loose rock, swelling ground, and quicksand. Workings of small cross section are driven almost invariably from one end and over the full cross section. Large-section workings are sometimes advanced in two stages, an advance heading in the top of the section and a bench that follows at a short interval. Long adits are sometimes driven from a number of faces by sinking shafts along the line of the adit and starting working faces in both directions from each shaft bottom. As an instance, four shafts were sunk on the line of the Sutro tunnel with the object of starting work on nine faces; but the shafts were abandoned, principally on account of the difficulty of draining them, and the work was prosecuted from the portal face alone.

Hard Rock.—Hand or machine drills are employed to drill successive rounds of holes in the face. Each round is blasted and the broken rock is removed. The section is trimmed to its clear dimensions by supplementary drilling and blasting. Machine drilling has superseded hand drilling to a large extent; the latter is now used only in prospecting or where local conditions prevent the employment of drilling machines. The layout and depth of the drill holes are determined by trial and are then used throughout the work, unless the formation changes radically. From two to three drills are employed on adits, one drill on ordinary drifts. As two or three shifts are operated, the cycle of operations is planned to conform as nearly as possible with the division into 8-hr. shifts. Thus drilling and blasting may be performed on one shift by a separate crew; mucking, track laying, and pipe laying on the second shift; and timbering on the third. In small drifts the drilling and mucking may be performed in one shift, with the cycle repeated in the next.

Soft Rock.—The method is the same in principle as the preceding, though fewer holes and less powder are required. The working is carried forward in successive rounds. Hand-held drills or unmounted air drills are frequently used in place of the bar- or column-mounted drills. Lighter drilling equipment is one conspicuous feature. Entries driven in coal are undercut or sheared by machine, and from two to four holes are used to break the remainder of the section. Timbering is usually required and is carried close to the face.

Loose Rock.—Rock which will not stand or which is unstable in advance of the timbering comes under this designation. There are all gradations, from sand to fractured rock. Where loose rock must be drilled and blasted, the rounds are made as short as possible, light powder charges are used, and the back is thoroughly barred down after blasting. Sufficient broken rock is removed to permit of placing the sets and top lagging boards. Where the back is too weak for this, the caps are provided with bridge pieces and the top lagging is driven in advance of the last set before mucking is started. In sand, gravel, or similar material blasting is unnecessary and forepoling is resorted to. The poling pieces are driven on both the top and sides of the section and serve to protect the miners from sudden falls or runs. The details of forepoling are given in the chapter on Support.

Swelling Ground.—The problem for material of this nature is one of support rather than excavation. The methods of supporting swelling ground are discussed in the chapter on Support. The unstable nature of the ground requires that the timbering or other methods of support be carried to the face. Such ground may stand unsupported a sufficient length of time to enable timbers to be placed.

Quicksand.—Sand saturated with water acts very much like a viscous fluid. Unlike swelling ground it almost instantly tends to run into a working and as a consequence must be thoroughly supported at all points. Forepoling with closely placed face boards and the use of bottom boards is required, and even this is frequently inadequate. Thorough draining of the sand is perhaps the best solution of the problem. This can be accomplished by resting the working until the water drains off or by driving pipes in advance of the face a sufficient distance to remove the water for a distance of 20 or 30 ft. The working can then be advanced by forepoling.

Cycle of Operations.—Where the formations penetrated are uniform, a systematic cycle of operations can be planned and the details perfected so as to attain rapid progress and economical construction. The sequential steps for hard, soft, and loose rock are given in the following outline.

A. *Hard Rock*
  1. Picking down loose rock
  2. Setting column or bar
  3. Pointing drills and connecting air hose
  4. Drilling
  5. Tearing down drills and blowing holes out with compressed air
  6. Charging holes
  7. Setting mucking plates; seldom used except for hand mucking
  8. Blasting
  9. Removing foul air at the face
  10. Mucking and picking down
  11. Timbering if necessary
  12. Track and pipe laying

B. *Soft Rock*
  1. Picking down back and loose rock at sides
  2. Undercutting face to depth of 6 ft. by machine
  3. Drilling three to five holes in the face with auger or jumper
  4. Charging and blasting
  5. Interval of time left for removal of gaseous products of blasting
  6. Moving cars up to face and cleaning up
  7. Mucking and transport

8. Timbering
9. Track and pipe laying; extending ventilating pipe

C. Loose Rock

1. Driving top boards
2. Partial excavation beneath top boards and at top of section
3. Driving top sideboards
4. Removal of upper breast boards
5. Excavation of upper half section
6. Placing of boom to support top boards
7. Driving of top boards to final position
8. Driving of sideboards and placing breast boards in advance of position; completing the excavation
9. Placing of new set
10. Removal of boom
11. Placing top boards in position for driving

Parallel Operations.—There is a sequence as in the work at the face indicated. Each step influences the final output or advance. A single miner and a drill can accomplish an amount measured by his skill and the conditions at the face. By doubling up, working two men and two drills, drilling and mucking time are cut in half. In the Carlton adit, five drills, a mechanical mucker, and a crew of 21 men made an average advance of 15.7 ft. per shift; in a 5- by 7-ft. development drift two miners, one drill, and hand mucking may accomplish 3 ft. per shift. Where time is of primary importance, increasing men and equipment at the face will give a higher rate of advance; but much depends upon sharp steel, good drills, and experienced men. The distance over which the broken rock is transported, the haulage speed, and efficiency in handling trains and cars are also factors. The outside crew takes care of tool sharpening and preparation of explosives, timber, and other supplies, as well as the conditioning of equipment and its repair.

The auto-feed drill, the drill carriage or jumbo, detachable bits, the mucking machine, the storage-battery locomotive, adequate ventilation and lighting, high-pressure air, modern explosives, and systematic organization of shift crews contribute to high rates of speed in adit, large drift, and crosscut operations. Timbering and the presence of much water slow down the rate. Other adverse factors, such as loose rock, swelling ground, and gas, greatly lower the rate. In long adits supplementary sidetracks or side drifts at regular intervals facilitate by-passing empty and loaded trains and provide storage places for the mucking machine and jumbo when they are not in use. The "cherry picker" lifts empty cars above or to the side of the track so that loaded cars can pass beneath or alongside. In the Carlton adit, stations for the jumbo were sliced out of the side at intervals of 3,000 ft. and the roof of the adit was slashed for the vertical car passer at intervals of 200 to 300 ft.

In single-heading work three shifts are employed, each shift crew drilling, blasting, mucking, and extending track and pipe for each round, the succeeding shift picking up the cycle in whatever stage it happens to be. Under adverse conditions two shifts may be used for the regular cycle of advance and a smaller shift crew for timbering and accessory work. In driving small development drifts a two-man crew completes a round along with track, pipe, and timbering. In tunnel operations where two headings are in proximity, greater labor efficiency is obtained by using two crews, one for drilling and blasting and the other for mucking and accessory work. When one face has been blasted the mucking crew enters and begins work while the drilling crew is at work on the alternate heading. When the distance between headings becomes such that too great an amount of time is required for the moving of equipment, single heading crews are used.

Table 113 gives time analyses for average rounds in driving mine adits; the Stanislaus tunnel is included for comparison. Table 114 gives some details of large mine adits.

TABLE 113.—TIME ANALYSES OF AVERAGE ROUNDS

	Average time, hr.				
	Ojuela, 1928	Carlton, 1941	Sheep Creek, 1912	Newhouse, 1902	Stanislaus, 1940
Setting up and drilling	2.33		4.21	10.0	3.20
Tearing down:					
Blowing Loading Blasting	0.58	1.77	0.40	2.5–3.5	0.30
Clearing gas	0.20		0.25		0.30
Cleaning up shovel	0.25				
Mucking	2.00	1.79	1.14	9	1.45
Interval before setup	0.25				0.36
Total time round, hr	5.61	3.58	6.00	22 (approx.)	5.61
Round depth, ft	8	8.22	4	8	7.61
Cu. ft. per round	800	853	320	1,800	883
Number of drills	4	5	4	3	5
Average feet per day	25	47.24	12	8.1	32.57

NOTE.—In the Sheep Creek example some overlap in time occurs; in the Stanislaus tunnel the single-heading data were taken; in the Newhouse example some estimation was made; in Ojuela, Carlton, and Stanislaus, drill carriages were used, in others column-drill mountings; in Sheep Creek and Newhouse hand mucking, in others mechanical mucking machines.

Equipment for Driving Adits and Drifts.—Prospecting and preliminary exploration equipment ranges from that used under pioneer conditions, such as the hand drill, hammer, sledge, pick, shovel, wheelbarrow, and blacksmith's outfit, to a more or less mechanized plant.

The minimum where electrical power is unavailable consists of light drills and mountings, drill rods and detachable bits, hand tools, timber-framing tools, a portable compressor of two or three light-drill capacity, pipe lines for air and water, a gasoline engine, blower and Ventube, light track, and several 12-cu. ft. cars, a magazine, and a blacksmith shop. Mules are used on long hauls. A service truck and accessory road may be necessary.

TABLE 114.—EXAMPLES OF LARGE MINE ADITS

	Carlton, Colorado	Metal-line, Wash.	Palidan-Suyoc, P.I.	Mahr, Peru	Elton, Utah	Ojuela, Mexico
Date	1939–41	1936	1937	1929	1937	1927–28
Width-height	8 × 9	8 × 8	11 × 11, I.T.	10 × 11.5	12 × 11	8 × 9
Length, ft	32,000	5,650	9,450	30,564	24,000	6,000
Rock	Granite	....	....	Volcanics	Alluvial; quartzite	Limestone
Rock drills, no.-type	5Af	3Af	4Pf	6Hf	Af	Drifters
In. dia	3½	3½	3½			
Mounting drill	5-jumbo	jumbo	4-jumbo	Hor. bar	5-jumbo	4-jumbo
Drilling time per round, hr.	1.57–1.86	....	1.5	2	....	1.83
Depth holes, ft	7.14 av.	....	....	9–11	....	8–10
Depth round, ft	5.93–8.45	....	8	9.8	9	8
Mucking time, hr	1.63–2.4M	S.L.	1.75M	2 S.L.	M	2M
Cu. yd. per ft	3.79	....	....	4.22		
Haulage	Sb. loco.	Sb. loco.	Diesel	....	Sb. loco.	Sb. loco.
Car size, cu ft	81	60	40	50	50	40
Ventilating pipe dia., in	18E	....	16 R.B.	23 R.B.	22 V raise	16 R.B.
Average monthly advance, ft.	1,421	422	....	978	....	714
Best month, ft	1,879	....	1,327	1,110	982	1,001
Average ft. per day	47.24	15.62	....	....	24	25
Men per day inside	63	14	78	....	39	54
Cost per ft	....	$23.58	$16.45	....	....	$21.37
Adverse conditions	....	....	....	Water	Water	
Reference: *Eng. Min. Jour.* for year	1941	1938	1938	1934	1940	1928

NOTE.—I.T. = inside timbers; Af = automatic feed; Pf = pneumatic feed; Hf = hand feed; 5-jumbo means 5 drills carried on jumbo; M = mechanical mucking machine; S.L. = slide loader; Sb. loc. = storage-battery locomotive; E = exhaust; R.B. = Root or Connersville blower; V raise = ventilating raise.

Long adits of large section require a portal plant consisting of power and compressor units and shops for tool repairs, tool sharpening, and timber framing. For work at the face 3½-in. drifter drills, drill carriage, mucking machine, storage-battery locomotive, two trains of muck cars, tracks, pipes, and electrical circuits are required. Ventilation requires a 15- to 18-in. ventilating pipe and blower. A magazine, water supply, and living quarters for the work crews are necessary as well as truck service and accessory roads. An operating mine provides equipment and facilities for all ordinary development and exploration work and need not be described here.

Labor Ratios.—Drifting and crosscut labor ratios vary widely and depend upon the experience of the miners, the quality of equipment, and the amount of timbering required. Under adverse conditions low labor ratios are obtained. With hand mucking an average of some 50 cu. ft. per man-shift may be obtained. Mechanical loading usually increases the labor ratio. Raises usually give higher labor ratios than drifts; shaft sinking, smaller ones. Table 115 gives a comparison of small-mine operations in which mechanical loading was not used and two tunnels in which mechanical mucking machines and jumbos were used.

TABLE 115.—CUBIC FEET PER MAN-SHIFT

	Drifts & crosscuts	Raises	Shafts	Stanislaus tunnel	Carlton adit
Rock (M.H. is medium hard)	Above M.H.	Same	Same	Granite	Granite
Cross section, ft.	5 × 7	4½ × 8	4½ × 9 inside timbers	9½ × 11¼	8 × 9
Area taken, sq. ft.	42	40	60	116.1	102.3
Crew per shift	2	2	2	26	21
Advance per shift, ft.	3	4	½	10.8	15.74
Cu. ft. per man-shift	63	80	15	48.3	76.2
Contract price per ft.	$6	$4	$35	$32.70	
Cost per cu. ft., cents	14.3	10	58	29.40	

NOTE.—Drifts, crosscuts, raises, and shafts were taken from a small mine and are for direct labor only; for drifts and crosscuts the men laid their own track and pipe, drilled, and hand-mucked; contract prices do not include supplies or power. At this same mine stoping was contracted at a labor price of 13 cents per cubic foot; station and shaft pockets at 12½ cents per cubic foot. The Stanislaus tunnel data were taken from a single heading operation.

Rate and Cost of Driving Adits, Drifts, etc.—The rate of driving depends on the hardness of the rock, the size of the section, the drilling appliances used, the air pressure, the kind and amount of powder, the skill and efficiency of the workers, the number of shifts per day, and the number of workers per shift. The most economical speed is that which gives the lowest unit cost. It is impossible to lay down hard and fast rules since each case must be adjusted to suit the individual conditions. However, in many cases the rate of driving is one round per shift, equipment and labor being adjusted to attain this end. The burn-cut round with its greater depth has increased shift footage, and mechanical muckers have made it possible to muck out long rounds in about the same time that was required for hand-mucking the shorter round. Cost per foot has been reduced in some instances but in others little advantage has accrued except speed in driving.

With hand-loading and ordinary depth of rounds, the rate of driving 5- × 7-ft. drifts may be 3 to 5 ft. per shift, with costs from $7 to $15 per foot. Mechanical muckers increase the per shift footage up to 7 ft. and in larger cross section up to 8 or 9 ft. In block-caving development, haulage drifts at Climax Molybdenum, 13 by 11 ft., were driven 9 ft. per shift, 4.5 ft. per man-shift; at Inspiration, 9 by 10 ft. to 7 by 8½ ft., footage was 5 ft. per shift, 1¼ ft. per man-shift; at Sunrise, 9- by 9-ft. drifts, 7 ft. per shift, 7 ft. per man-shift (from Bucky). Costs are not available. With power shovels costs may range from 10 to 15 cents per cubic foot; with hand-mucking 15 to 30 cents per cubic foot.

Raises are extended one round per shift with round depths from 4 to 7 ft. Costs vary with the cross section and the hardness of the rock. In hard rocks the range may be from 18 to 30 cents per cubic foot.

Delineating ore limits by diamond drilling gives shift footages of 10 to 20 ft. and costs of $1 to $1.50 per foot.

TABLE 116.—LABOR, POWER, AND SUPPLY UNITS—DEVELOPMENT IN ROCK AND ORE, MONTREAL MINE, MONTREAL, WIS.

Dimensions	Physical prop.	Man-hours per foot				Explosives, lb. per ft.	Timber, bd. ft.
		Breaking, drilling, blasting	Timbering	Mucking	Total labor		
Rock development:							
Main-level drift and cross-cuts 9 by 9 ft.	Med. hard	4.89	3.07	3.15	11.11	17.6	64
8 by 8 ft.	Hard	6.22	....	2.67	8.89	19.35	
Sublevel drift and cross-cuts 5 by 8 ft.	Med. hard	3.38	2.18	2.39	7.95	9.67	37
5 by 7 ft.	Hard	4.54	....	1.95	6.49	9.3	
Raising, 2 compt. 4 by 8 ft.	Hard	7.09	5.94	....	13.03	13.67	60
4 by 5 ft.	Med. hard	6.38	....	....	6.38	8.61	
Ore development:							
Main-level drift and cross-cuts 9 by 9 ft.	Med. soft	4.03	3.25	3.22	10.50	16.3	64
Sublevel 5 by 8 ft.	Med. soft	1.56	1.39	1.97	4.92	6.0	37
Sublevel 5 by 7 ft.	Med. hard	1.83	....	2.02	3.85	6.2	
Raising, 2 compt. 4 by 8 ft.	Med. soft	4.16	4.00	....	8.16	6.1	60
Single compt. 4 by 4 ft.	Med. soft	2.60	2.23	....	4.83	5.6	30
4 by 5 ft.	Med. soft	3.23	....	....	3.23	6.0	
Total, all development	....	2.88	1.00	1.65	7.58	8.4	15

NOTE.—Haulage and hoisting man-hours per foot 1.58; supervision 0.47. Total power, kw.-hr. per foot: Air compression 37, hoisting 22, haulage 3.4. Labor including supervision 62 per cent of total cost; power and supplies; drill steel 38 per cent of total cost.

From *Inf. Circ.* 6369, *U.S. Bur. Mines.*

Most mining companies contract development work at a given price per foot. Power and tools are usually supplied and contractors purchase their own explosives. The following example of the Alaska Juneau Co.'s practice will serve to give costs for hard-rock driving: Main haulageways, 9 by 9 ft., $12.50 to $14.50 per foot; drifts and crosscuts, 7 by 5 ft., $7 to $8.50; powder drifts, 4 by 3 ft., $5.50 to $6.50; stope raises, 5 by 5 ft.,

$6.50 to $8.50; oreway raises or transfers, 7 by 7 ft., $10 to $12 per foot. The contract costs given are for direct labor and explosives, which amount to approximately 60 per cent of the total cost per foot. An example is given in Table 116.

DEVELOPMENT COSTS AND FOOTAGES

Finished Development Represents Money Expended.—As was indicated at the beginning of this chapter, to develop completely a large ore body in advance of mining operations would represent a large expenditure. While this would be desirable from the standpoint of simplifying and making more definite the mining problem, it is not expedient for economic reasons. It has the further objection that more or less expense would be incurred in keeping the development workings open. The development is consequently planned so that it will be well in advance of mining operations. Just sufficient new development to ensure a steady production of ore represents the ideal toward which the engineer should aim. It is impossible to lay down any hard and fast rules, and the good judgment of the engineer would decide in any one case. In most metal mines it is desirable to work several levels simultaneously, thus obtaining a steady production of a more or less uniform grade. Ores vary in richnesss in different parts of a vein and it is undesirable to mine all the ore from a rich stope and then the lean ores from other parts. A judicious selection of rich and poor ores is better, and a more uniform metal production can be made. The engineer should endeavor to place the smallest amount of capital into advance development work consistent with the continuous production of a given tonnage.

Maximum and Minimum Development Costs[a]

Mining method	Tons per ft.	Development cost per ton	Mining cost per ton	Development, per cent of mining
Glory or mill hole	40–220	$0.053–$0.35	$0.64–$1.83	8–29
Open stope, iron mines, per long ton	30–125	0.103– 0.25	0.93– 1.41	9–35
Open stope, various mines	11–70	0.197– 1.16	1.12– 5.01	10–29
Shrinkage stoping	5–210	0.080– 2.87	0.30– 6.76	14–42
Cut-and-fill stoping	7–30	0.490– 0.92	2.44– 6.42	5–28
Square-set stoping	6–60	0.131– 2.09	1.77– 7.75	4–35
Top slice, per long ton	34–200	0.050– 0.25	1.10– 2.30	5–18
Block-caving method	70–280	0.046– 0.30	0.40– 1.32	10–29
Sublevel caving, per long ton	50–60	0.254– 0.32	1.34– 1.54	19–20

[a] Compiled from tables by M. S. Elsing, *Eng. Min. Jour.*, vol. 137, p. 245.

Development costs theoretically should be distributed over the tonnage developed by the workings. This is sometimes done, but in practice the most convenient method is to add the current development costs to the mining costs or distribute them over the tonnage produced in the given time. In the accompanying table maximum and minimum mining and development costs are compared for different mining methods.

The preliminary development at Ajo (Arizona) is of sufficient interest to be included.

Development at Ajo (Arizona) by the Calumet & Arizona Mining Co.[a]

Diamond-drill holes (84)	23,097 ft.	
Test pits (77)	3,955 ft.	
Drifting	1,513 ft.	
Sinking and raising on drill holes	317 ft.	
Ore estimate:		
Carbonate ore	11,954,000 tons	1.54 per cent copper
Sulphide ore	28,303,000 tons	1.50 per cent copper
Total	40,258,000 tons	1.51 per cent copper

Time required, approximately 2 years.

[a] *Trans. A.I.M.E.*, vol. 49, p. 606.

The preliminary development work required for iron-ore deposits upon the Cayuna Range, Minn., is stated by P. W. Donovan[1] to consist of from 12 to 15 holes of an approximate average depth of 260 ft. each or 3,400 ft. of drilling per 40-acre block. The holes are placed in cross sections 300 ft. apart along the strike of the ore leases. In six developed properties upon this range an average of 3,000 ft. of drilling was required per 40-acre block. Each foot of drill hole developed 250 tons of mer-

Table 117.—Annual Development Footages, Alaska Juneau Mine

Detail, ft.	1935	1936	1937	1938	Cumulative 1911–38
Drifts, crosscuts	3,926	8,297	13,912	17,295	149,452
Raises	6,676	10,916	10,178	14,637	137,927
Winzes	34	635	281	134	4,875
Intermediate drifts	832	248	1,074	230	13,555
Powder drifts	1,472	2,463	5,484	6,744	66,021
Bulldozing chambers	622	707	208	247	3,856
Stations	22	208	129	129	5,586
Total footage	13,584	23,474	31,266	39,416	381,272
Stope area cut out, sq. ft.	27,608	81,792	82,350	58,308	1,236,987
Tons milled	2,091,475	2,462,046	2,251,079	2,478,928	37,530,918
Tons rejected	1,638,185	1,904,754	2,191,681	2,184,952	32,596,720

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[1] Some Aspects of Exploration and Drilling on the Cayuna Range. P. W. Donovan, *Trans. Lake Superior Min. Inst.*, September, 1915.

chantable ore. The cost for preliminary development approximated 1 cent per ton.

The footage in development required in a given mine ranges in practice between wide extremes. The development plan, the size of the ore bodies, and the liberality of advance development are factors that vary from mine to mine. Certain figures are given in Table 117 showing the annual development footages in the case of the Alaska Juneau mine.

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CHAPTER XV

UNDERGROUND METHODS

Nomenclature.—By the term "method," as applied to the winning of ore, is meant all the operations involved in excavation and support and the handling of ore, waste, men, supplies, and air. Experience has evolved certain definite methods, which have received names derived from some peculiar characteristic of the method. Necessarily the nomenclature is not consistent, nor is it clear from the name of the method just what is included. Names are used in some localities with a somewhat different sense from that in other mining districts. As to excavation, all methods involve one or several of the following: Underhand stoping, overhand stoping, rill stoping, caving, and undercutting. As to support, all methods involve one or several of the following: Unsupported or open stopes, stull-timbered stopes, pillar-supported stopes, waste-filled stopes, ore-filled stopes (shrinkage stopes), square-set stopes, and square-set and filled stopes. As to sequence in the removal of the ore, methods may involve the following: Top slicing, bottom slicing, side slicing, block-caving, or room-and-pillar work. It is evident that a comprehensive nomenclature would be exceedingly clumsy, and as a consequence the terms ordinarily in use are retained, with but few exceptions, in this treatise. The methods discussed are:

1. Underhand stoping
2. Overhand stoping
    - Open stopes
    - Filled stopes
3. Combined overhand and underhand stoping
4. Square-set stoping
    - Open stopes
    - Partially filled stopes
    - Filled stopes
    - Alternate pillar and stope
5. Shrinkage stoping
    - Without ore passes
    - With ore passes
    - Alternate pillar and stope
    - (*a*) With subsequent fill
    - (*b*) With backfill
6. Top slicing and cover caving
    - By drifts
    - By rooms
7. Combined top slicing and shrinkage stoping
8. Top slicing with partial ore caving
9. Block-caving; undercut caving
10. Shrinkage stoping and block-caving

11. Longwall
12. Room and pillar
13. Bord and pillar
14. Special methods

DEFINITIONS

Stope.—Any excavation underground, other than development workings, made for the purpose of removing ore. The outlines of a stope are determined by the outlines of the ore body.

Stoping is the act of removing the ore. It includes all the operations incident to the excavation of the material.

Back is the lower or under surface of a block of ore. It is a surface usually horizontal from end to end, but may be inclined from side to side depending upon the dip of the vein. The portion of an ore body between two levels is called the back.

Breast is the vertical end surface of a block.

Face may be applied to almost any flat surface, although usually a vertical surface is implied.

Stope Drive.—This is a working similar to a drift and is the first face driven in opening up a stope.

Ore Block.—A section of a vein bounded on top and bottom by upper and lower drifts and on one or both ends by winzes or raises and ready for stoping is called a block.

Slice.—In an ore body of considerable lateral extent and thickness the ore is removed in layers termed slices. A slice may be 6, 12, 20, or 40 ft. thick.

Ore Pass or Chute.—Vertical or inclined passageways for the downward movement of ore are called ore passes or ore chutes. They connect with the lower level and are equipped with gates or other appliances for controlling the flow of ore. A "transfer" is driven in ore or country rock and is used for transfering ore from an upper to a lower part of the workings.

Manway.—Passages, vertical or inclined, for the accommodation of ladders, pipes, and timber chutes are called manways, winzes, or in some cases raises. Their purpose is to give convenient access to the stope.

Underhand Stoping.—The bench is broken by drill holes drilled directly downward or at a steep angle, and the ore is broken out at the end of a block. Excavation proceeds from the top toward the bottom of a block.

Overhand Stoping.—The bench is broken by drill holes, which are drilled vertically upward in a back, horizontally, or at an angle in a breast. The ore is broken so as to fall downward. Excavation proceeds from below upward.

Rill Stoping.—The face drilled is neither vertical nor horizontal but inclined at an angle to the drift, and the ore is broken so as to fall downward. Excavation proceeds from below upward.

DESCRIPTION OF METHODS

1. Underhand Stoping.—The stope is started from a winze or connection between an upper and lower drift. A horizontal slice from 6 to 8 ft. high is started on the top of an ore block. The broken ore falls into the ore pass in the winze. When the slice has reached a point where the ore must be shoveled to the ore pass, another slice is started immediately below the top slice. This procedure is continued until the slices reach a limit beyond the service of the ore pass. This limit for a vertical vein is established by a plane of approximately 40 deg. dip, the lower end of the plane intersecting the lower end of the ore pass and the upper end being approximately 100 ft. from the top of the ore pass. For a vein of 40 deg. dip the vertical plane perpendicular to the strike at the ore pass is the limit of service by the ore pass. Forty degrees may be taken as the limit angle of slope necessary for the downward movement of ore with

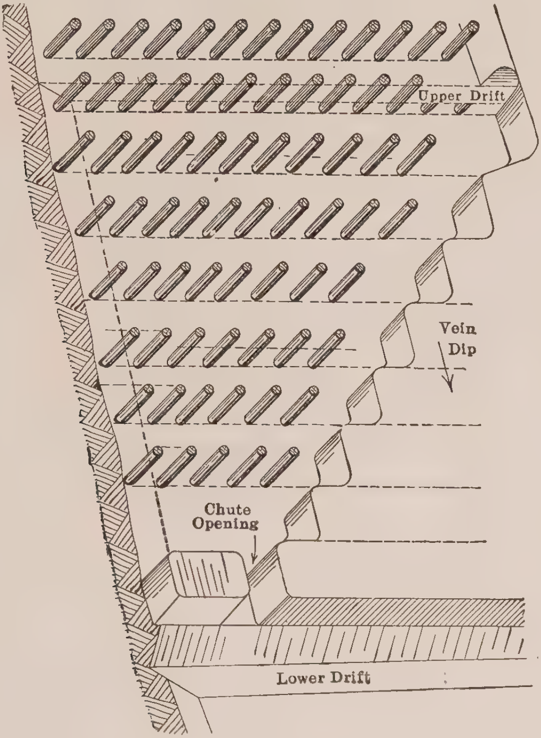


FIG. 248.—Underhand stoping.

but a moderate amount of shoveling. The function of the ore pass is served as long as the ore gravitates to it. More or less shoveling is always required. The contour of the ore faces approximates the 40-deg. slope or a steeper one so as to reduce shoveling to a minimum. It is customary to open new ore passes as the stope is extended along the drift (Fig. 248).

Support of the side wall is effected where necessary by cutting hitches and placing stulls in position. With good walls the stulls are placed irregularly and only at points where the hanging wall is weak. With poor walls stulls are systematically placed in horizontal rows 6 to 8 ft. apart vertically and the same distance horizontally.

Waste occurring in moderate quantity is stowed on platforms that are supported upon stulls. Where 20 per cent of the volume of ore is waste, the broken waste will occupy 30 per cent of the volume of the stope (assuming 1 ton of solid is 13 cu. ft. and 1 ton of loose ore 20 cu. ft.), and it is therefore evident that only a moderate proportion of waste can be handled. Its separation can be only imperfectly accomplished in the stope.

Access to the stope is through manways constructed alongside an ore pass or from the upper level by ladders. Ventilation is easily effected, as the winze gives connection between the levels and the stope is open except where waste packs are placed, and these do not interfere with the circulation of the air. Compressed-air pipes are tapped from the level above and air hose are easily placed where desired.

The Advantages of the Method.—When down holes are drilled, the workers are on top of the ore, a position safer than where they work below a freshly broken back; little timber is required where the walls are solid and do not flake off; loose rock can be safely barred down; fine ore can be readily worked down and there is practically no loss from fines; the method can be used for the winning of ore below a given level without the necessity of driving a lower level; it is especially suitable for the working of ore bodies which are in an approximately vertical position or which dip at a steep angle.

The Disadvantage of the Method.—A considerable amount of shoveling is required; the broken ore from the upper slices interferes with operations on the lower; little ore can be allowed to remain in the stopes; it is impracticable on veins of low dip, unless cheap labor is available for shoveling; veins containing much waste can be won by more economical methods; hoisting of the ore is necessary where a stope is worked below a level.

Modifications of Underhand Stoping.—Sheet ground in the Joplin district, Mo., is mined by a modification of underhand stoping. The ore bodies, which are flat and usually less than 20 to 30 ft. in thickness, are opened by shafts from 100 to 300 ft. deep. The extraction is accomplished from one level. A heading 6 to 8 ft. high is driven under the roof of the deposit, and the remaining bench is removed by drilling holes vertically downward from the heading or by deep downwardly inclined holes in the base of the bench. Practically no timbers are used, as the roof is supported by pillars irregularly placed at intervals of 25 to 50 ft. Figure 249 shows the appearance of the underground excavation. Ore transport is readily effected by tracks from the working face to the shaft. A pillar of ore is left about the shaft. Only the presence of a comparatively firm roof permits the use of such a simple method.

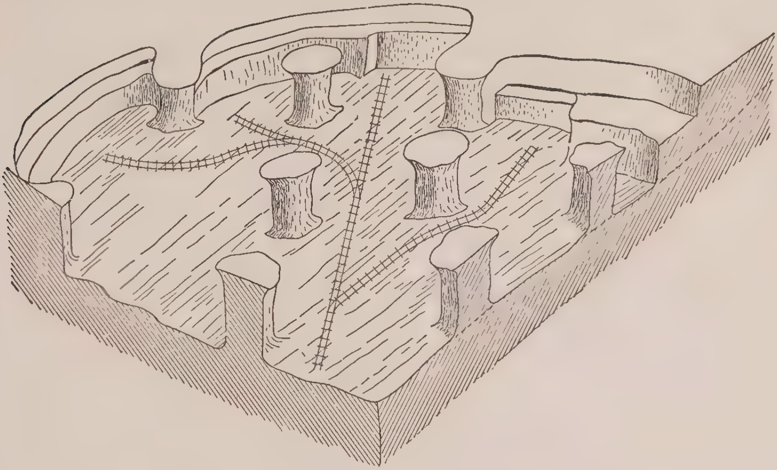


Fig. 249.—Method of mining sheet ground in the Joplin district.

In the iron mines of Michigan a similar method is followed with the difference that the stope is narrow (the width of the ore) and long. Stoping is started from a raise at one end of the section to be attacked. An advance heading is run, and from this the ore below to the floor of the level is removed by underhand stoping. A pillar of ore protects the top of the stope from the next level. It is obvious that only a hard ore and good walls would permit the use of this method. Figure 250 illustrates the method.

The method of mining potash salts at Stassfurt, Germany, is illustrated in Fig. 251. The potash formation is firm and enclosed by a hanging wall of anhydrite, which stands well over wide chambers, and a footwall of rock salt. The potash salts are mined in slices each 25 ft. thick. On a given level operations start from a footwall drift. Chambers are opened out from crosscuts. Between every pair of chambers a 50-ft. rib or pillar is left, while between the chambers of a pair a 25-ft. rib is left. The chamber, 25 ft. in height, is opened out by an advance

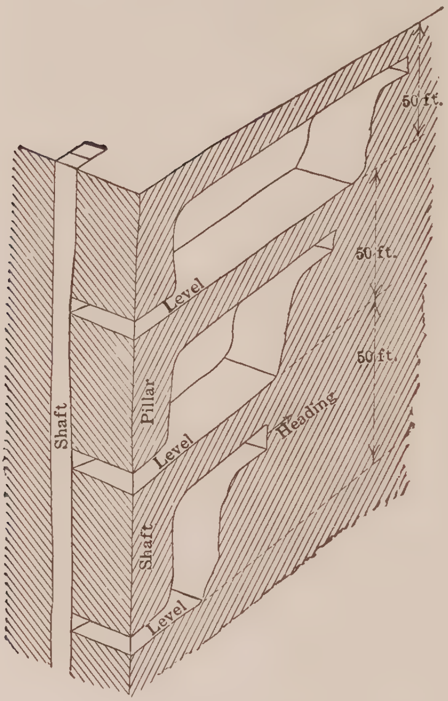


FIG. 250.—Underhand stoping of iron ores.

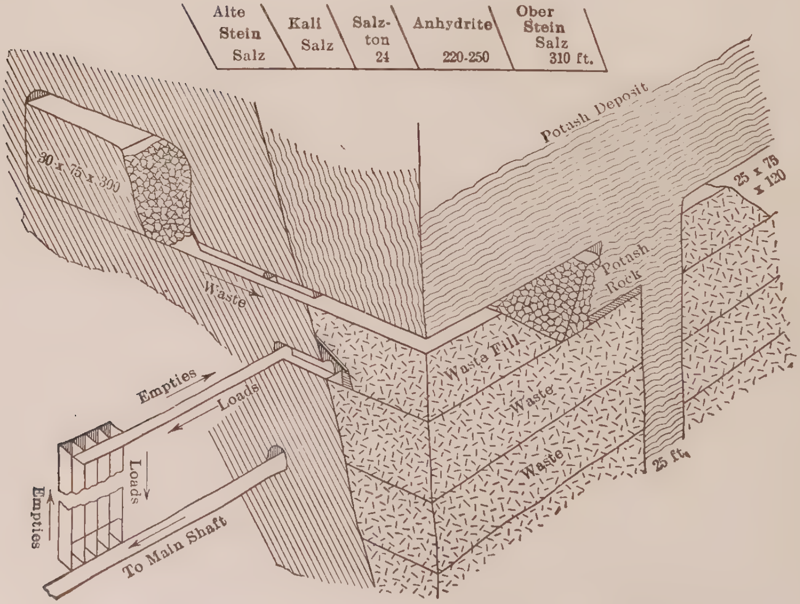
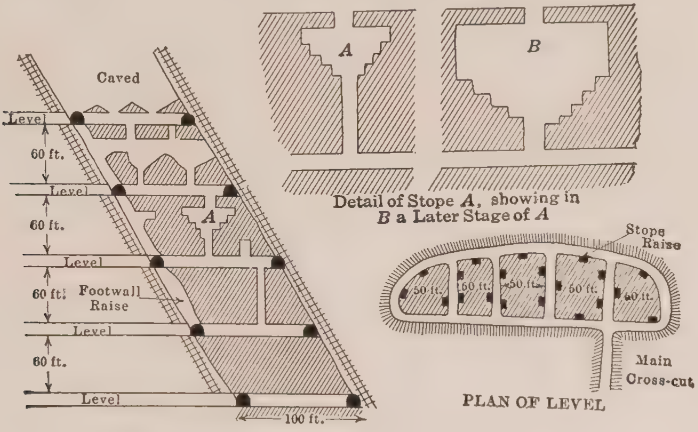


FIG. 251.—Mining salt deposits at Stassford.

heading in the top, and the bench below is stoped underhand or by horizontal holes. As soon as one end of the chamber is cleared, the drift on the footwall side is protected by waste walls and timber, and filling is dumped in from a crosscut that enters at the top of the chamber. The filling follows closely upon the excavation of the potash. Filling is obtained from chambers that crosscut the footwall. As soon as a chamber is filled, operations begin on the next 25-ft. section above. No timber is used, although the width of the unsupported chamber may be from 50 to 75 ft. Operations continue upward until within a short distance of the next main level, where a transverse pillar is left. The ribs extend in an unbroken vertical line between the chambers.



TRANSVERSE SECTION

FIG. 252.—Mining iron ore. (*Trans. L.S.M.I.*)

Another method used in the iron mines of Michigan has certain features resembling the glory-hole method used in surface work. Raises are driven from a lower level and terminate at a point a sufficient distance below the upper level to leave a pillar to protect the stope. The stope is then opened out on either side of the raise. Pillars or ribs are left between neighboring stopes. The ore is worked down to the level, the raise serving as a chute. The bottom of the stope is sloped so as to enable the ore to be worked down with a minimum of shoveling. The stope raise connects with the upper level and affords access and ventilation. The stopes intersect or a thin rib is left between. Ribs and pillars eventually cave, and the ore is recovered by drawing it off through footwall raises. Figure 252 illustrates the method as applied in Mine 21[1] on the Marquette Range, Mich. The method is applied where no timbering is required to support the stope.

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[1] Mining Methods on the Marquette Range. *Trans. Lake Superior Min. Inst.*, vol. 19, p. 131.

2. Overhand Stoping.—The nomenclature of this method is more or less involved, and terms are used that are variously applied. Part of the confusion arises from the fact that there are five possible methods of extending the benches used to facilitate breaking. Two methods are shown in Fig. 253*a*. In the first, horizontal slices are taken off parallel with the line of the drift. The horizontal slices may be taken off in sequence, or succeeding slices may follow the first with an interval of greater or less extent between them. The inverted stepped appearance is obtained when a short distance intervenes between the stope faces. To this method the term "back stoping" is commonly applied. Where the dip of the vein is low and the slices are carried in sequence instead of simultaneously, the term "longwall stoping" is sometimes used. It is

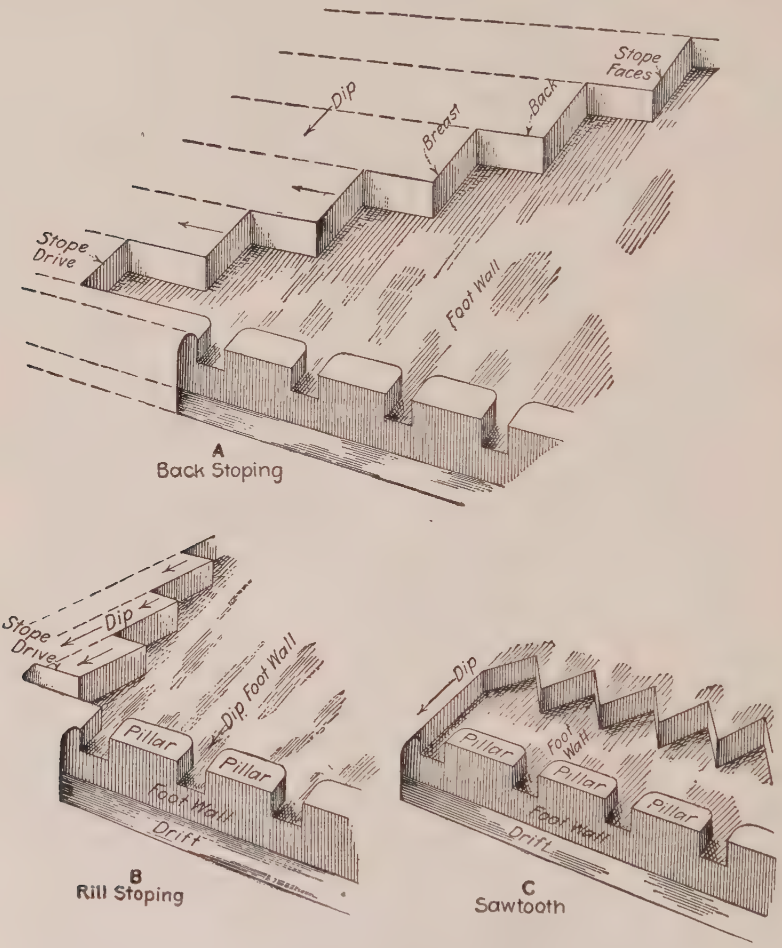

FIG. 253.—Overhand stoping.

customary to term the first or lowest slice the "stope drive," the "cutting-out stope," or "lead stope." This may be driven directly from the drift, with the drift protected by stulls and heavy lagging, or it may be driven with a pillar interval between the drift and the stope drive. The second method is to drive the stope parallel with a raise or up the dip. The slices may be driven sequentially or stepped, as in the former case. The method is referred to as "side stoping," "breast stoping," and sometimes "raise stoping." It is of uncommon occurrence except in very flat veins, and even here it has no particular advantages over back stoping. The term "breast stoping" is more commonly applied to the excavation of a horizontal slice on the floor of a level in a wide vein. Figure 253*b* illustrates rill stoping. In this method the stope face is inclined to the drift. The slices may be extended from the bottom up or from the top down. Figure 253*c* illustrates what is termed "sawtooth stoping." In this the general line of advance is up the dip. The benches are advanced in a line parallel with the drift. The method permits a large number of machines to be operated but requires the miners to work under a comparatively dangerous back.

Overhand stopes may be open or filled. In open stopes where the dip of the vein exceeds 30 deg., stulls have to be placed at regular intervals in order to support platforms from which the miners work. They also serve to support the walls. With good walls, stulls from 18 to 24 ft. and of moderate diameter, 12 to 14 in., can be used. Where walls are heavy the maximum stoping width is reduced to 15 or 16 ft., and heavy stulls have to be used. In steep veins the stulls are placed horizontally and reinforced with vertical props. This is an approximation to square-set stoping, which is described in another section. The arrangement permits a wider stope. In steep veins the ore is broken down upon floors supported by stulls and then shoveled down, but with flatter veins it can be broken down upon the footwall and allowed to slide to the bottom of the stope. More or less ore is allowed to accumulate in the bottom of the stope, with the object of protecting the drift timbers and chutes.

The open-stope method applies in general to narrow veins. The minimum width varies between 3 and 4 ft., and as a consequence narrower veins require the removal of more or less wall rock and the sorting of the ore from the waste. Where the stopes are filled, the waste for filling may sometimes be sorted out of the ore or secured from a source without the stope. Where part of the ore is allowed to remain in the stope as a means of support and to afford a working platform, the method is termed "shrinkage stoping" and is described under that title. Filling may be placed in horizontal layers, the layer of filling following the removal of a slice of ore. Another slice of ore and another layer of filling follow in sequence. This operation is continued to a distance about 20 ft. from the next level. This pillar can be removed by square setting or by stoping with an inclined back and inclined fill and retreating from one end of the stope to the other. The former method is used on wide viens, the latter on narrow. The method as described above is suitable for a wide or narrow ore body where the walls are good and the back stands without support. With a moderately strong back and a wide ore body, temporary support by means of timber cribs can be placed on the fill and the back can be supported. Figure 254 illustrates a typical case of a filled stope.

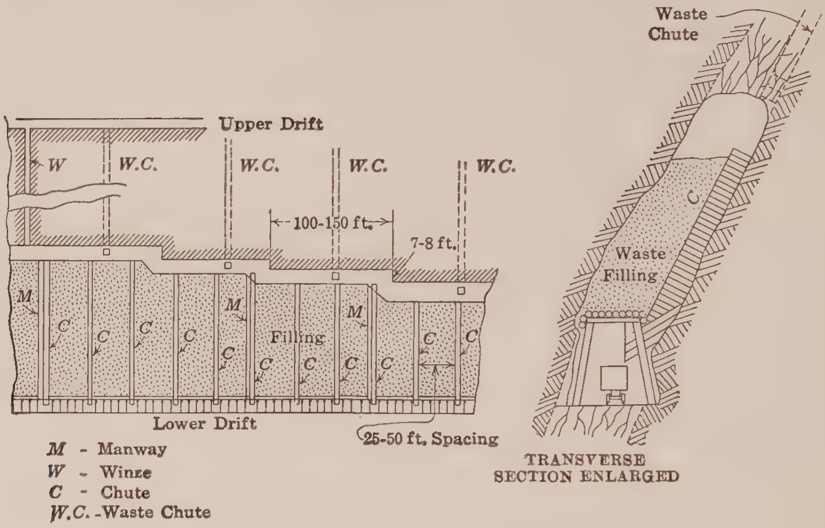

FIG. 254.—Overhand stoping with fill.

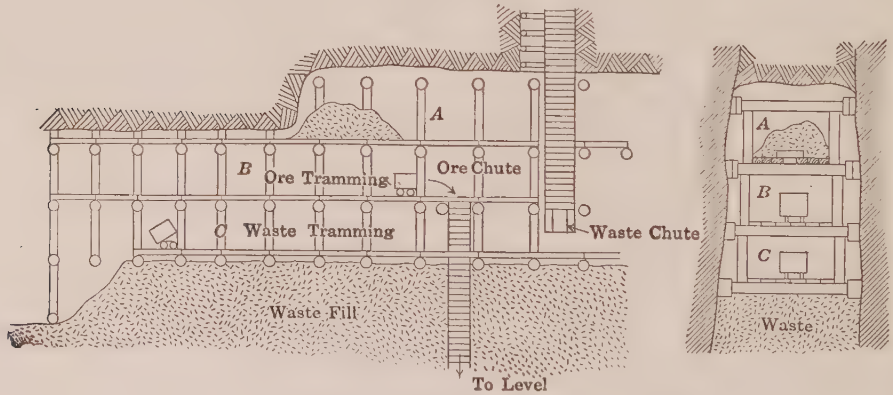

FIG. 255.—Overhand stoping with stulls and waste fill.

In stull-timbered stopes on vertical veins where filling is employed, the operations of breaking, ore handling, and waste handling can be segregated on three separate floors, as shown in Fig. 255. Floor *A* is the ore-breaking floor. The ore is broken down upon a heavy floor supported by stulls. In the center of the floor a 12-in. plank is omitted and small, short boards are used to cover the opening, which extends parallel with the length of the stope. By removing the short crossboards the pile of broken ore can be run through the central opening into the car below. Floor *B* is the ore-tramming floor. The ore cars are trammed to the chutes, which are spaced 50 to 75 ft. apart and discharge on the level below. Floor *C* is kept open and used for tramming waste. The waste chute extends to the level above, and the car is loaded at the chute and trammed to the fill. Waste chutes can be spaced at 100- to 150-ft. intervals. The method involves a minimum of shoveling.

With a comparatively weak back and wide ore body, transverse stoping instead of longitudinal (extending parallel to the length of the ore body) can be resorted to. In this method each horizontal slice is worked off in narrow stopes extending from wall to wall. Each stope is filled before its neighbor is removed. The method is termed "transverse back stoping and filling," while the former is termed "longitudinal back stoping and filling."

A method essentially the same as transverse back stoping is illustrated in Fig. 256. A stope 300 ft. long and 20 ft. wide is represented. The ore body was more or less broken because a stope that had been started by

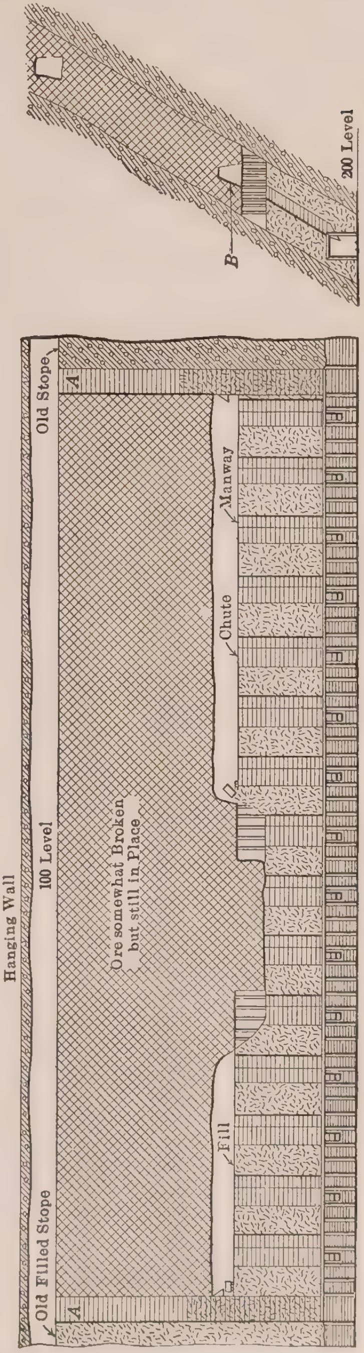

FIG. 256.—Transverse back stoping. (*Eng. Min. Jour.*)

the square-set system was lost by a fire, which caused the stope to cave. An upper and lower level were driven and connected by rises at *A* and *A*. The lower level was timbered, and chutes and manways were constructed at 16-ft. intervals. The end raises were used for waste chutes. The ore was mined in rooms 10 ft. high, each centered on a chute extending to limits midway between the chutes and from wall to wall. The rooms were mined sequentially from both ends of the stope. The heavy back was supported by tapered props and headboards. Each room was filled from a center drift shown at *B*, which was extended with the mining. After a room was completed, the chute and manway were built up to the back and the room was filled. In working the next 10-ft. lift, the tapered props in the fill beneath were pulled by means of a hydraulic jack and reused.[1]

Incline filling, drawn from a chute, is used whenever possible instead of horizontal layers, which require the expenditure of considerable labor in placing the filling. Incline filling is suitable for relatively narrow ore bodies. It is applied by starting a stope at a raise and using a steeply stepped stope face or rill stoping. The waste is run in from a waste chute from which the stope starts. Plank flooring is placed on the waste and another slice of ore is broken down, falling upon the floor and sliding down to a chute. The floor is then removed and waste is run in. The angle of fill and stope is approximately 40 deg. The term "incline cut and fill" is sometimes applied.

In the Butte district a combined shrinkage and filling method is in use where the back is strong enough to support itself across the width of the ore body. In this method the miners stand on the broken ore and reach the face. The pile of broken ore is advanced as the face advances (the face is inclined as in incline cut and fill). The rear of the pile of broken ore is shoveled into chutes, which are extended up through the fill. The waste is dumped in after the broken ore is removed. The waste pile is advanced as the pile of broken ore retreats. The next slice or lift of ore starts on the pile of waste and is mined out in the same manner. The term "backfilling" is applied to the method.

Overhand stoping is preferably started from a winze, although it may be started from a raise stope. The latter case is objectionable on account of difficulty of proper ventilation. Stopes may be started in both directions from the winzes. The winze serves as a manway, a pipeway, and a means for the lowering of timber and supplies. If waste is to be brought to the stope, a two-compartment winze is constructed, one compartment being used as a waste chute. The vertical height of the bench taken varies from 6 to 9 ft. In stopes where the dip exceeds 40 deg. the ore gravitates to the bottom, from which it is drawn off through chutes. Chutes are spaced from 25 to 30 ft. on narrow veins and from 50 to 60 ft. on wide veins. In some cases inclined plank flooring, supported by the stulls and forming a hopperlike bottom to the stope, is installed. In waste-filled stopes where the waste is placed in horizontal layers, the ore is broken down upon plank floors, the waste is sorted out, and the ore is loaded into cars and wheelbarrows and trammed to the nearest chute. In this case the chutes are extended up through the waste fill and are often spaced at 50-ft. centers along the center line of the stope. For convenient shoveling the spacing of chutes is about 25 ft., center to center. Close spacing of chutes leaves a smaller residuum of ore in the stope and avoids the use of the wheelbarrow or car in waste-filled stopes. In veins of 30- to 40-deg. dip, wooden or metal chutes are used for moving the ore from the face to the drift. The chutes are moved as the faces advance. In this case stope faces are stepped only slightly in advance of each other. For veins of flatter dip, mechanical means of transport are used. The most common appliances are the shaking chute and the gravity plane. With the gravity plane the stope faces are extended sequentially in a line parallel with the drift. An auxiliary track follows the stope face as it is extended, and the cars are moved on this to the plane down which they are lowered.

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[1] Description taken from Mining Ore from a Caved Stope. J. E. Harding, *Eng. Min. Jour.*, July 10, 1915, p. 71.

See for other examples of transverse back stoping:

Extraction of Ore from Wide Veins or Masses. G. D. Delprat, *Trans. A.I.M.E.*, vol. 21, p. 89.

The Chapin Iron Mine. Per Larsson, *Trans. A.I.M.E.*, vol. 16, p. 122.

Overhand stoping in narrow veins from 20- to 30-deg. dip, or flatter, necessitates a different procedure from the chute-served stope. At the North Star mine a raise is extended from the lower to the upper level as nearly as possible in the center of the ore shoot. In driving the raise the broken rock is moved by a scraper and a tail-rope hoist to the lower level, where it is loaded into cars. The raise on completion is provided with a car dump at *D* in Fig. 257 and is equipped with a double track. The stope is extended up the dip in successive cuts of some 30 to 60 ft., measured along the dip. The stope faces are served by a track, which is placed in a shallow bench blasted out of the footwall. A row of stulls and boards confines the broken ore and prevents it from being scattered down the footwall. The ore is loaded either by hand or by scrapers into cars on the stope track, and these are trammed to the head of the gravity incline at the head of which is a slick sheet, *B*, where they are lowered to the dump at the lower level and transferred to cars on the level. Empty cars are hauled up by the loaded cars. Waste rock is thrown back of the boards. Scrapers are used for a short distance in the lower part of the stope. After the scraper limit is reached, the gravity-incline tracks are used. The incline is served by a go-devil, a three-sheave device with a control brake, which is attached to a heavy stull at A, as shown in Fig. 257, and moved into a new position for each successive advance up the dip. Levels are 300 ft. apart on the dip.

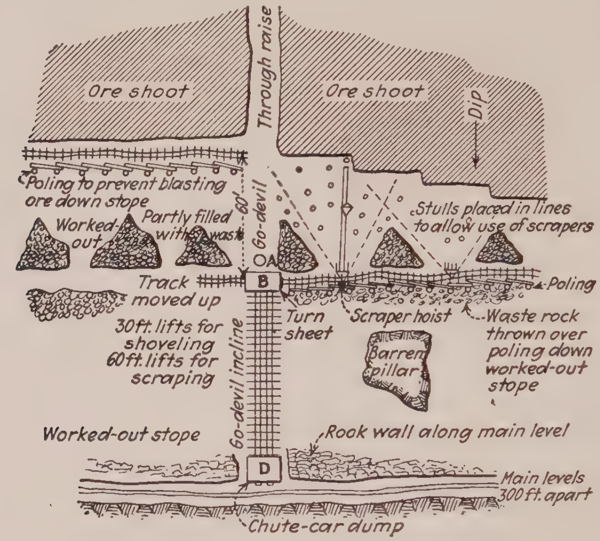

Ore shoot
Through raise
Ore shoot
Dip
Poling to prevent blasting ore down stope
Worked-out
Partly filled with waste
60'
Go-devil
A
Stulls placed in lines to allow use of scrapers
B
Track moved up
Poling
Turn sheet
Scraper hoist
Waste rock thrown over poling down worked-out stope
30ft. lifts for shoveling 60ft. lifts for scraping
Go-devil incline
Barren pillar
Worked-out stope
Rook wall along main level
D
Main levels 300 ft. apart
Chute-car dump

FIG. 257.—Flat overhand stoping with gravity incline and auxiliary scrapers. (*J. A. Fulton and A. B. Foote.*)

Ore bodies of greater thickness than 4 or 5 ft. admit of working in a similar manner where the ore is moderate to low grade. An alternative arrangement is the serving of the stope area by a system of footwall drifts, from which inclined raises are extended to the footwall of the stope. These are used as transfers and for chute loading. Scrapers are used in the stope to move the ore to the transfers, which are so distributed as to provide a maximum distance of from 75 to 150 ft. The footwall development of drifts and raises increases the development footage but avoids the use of gravity inclines and tracks in the stopes.

In the deeper gold mines of South Africa, stoping practice is influenced greatly by the action of the hanging wall and the necessity for supporting it securely close to the working faces. Several different methods, two of which will be described, have been evolved for overhand stoping at great depths. Stopes are of great areal extent, the stoped portion being from 4 to 5 ft. in thickness and the dip variable from 15 to 40 deg. The support of the hanging wall in the upper portions of the "reef" was by stulls and ore pillars. As greater depths were reached, pigsties took the place of stulls, and these were placed closer and closer together as the difficulty of supporting the hanging wall increased with depth. To a considerable extent sand filling was practiced, which served to give a greater control over hanging-wall movement; but with the greater depths the problem of support in the near vicinity of the working faces was solved by a modified form of shrinkage stoping, in which the broken ore was stowed in the open spaces and retrieved at a later time, closely spaced pigsties finally taking the place of the broken ore.

The following description of the present practice is taken from an article by Paul Selby. After a winze raise in a given stoping area is connected, the bottom of the stope is cut out, the stope face making an angle of about 35 deg. with the dip. As soon as the faces have advanced sufficiently, stulls are placed on either side of the winze and walls are built up on these, the space between being filled with broken ore, which at first is mucked in and afterwards is filled by the blast. The pack is kept close to the face, and only the surplus ore is removed. The packs cover areas 24 ft. along the strike and from 40 to 60 ft. up the dip, horizontal open spaces being left some 6 ft. in width along the strike at regular intervals. Similar passages are also left parallel to the dip, their widths varying from 4 to 6 ft. The plan of a stope is shown in Fig. 258. The percentage of area packed with ore and waste rock varies from 40 to 60 per cent. On the basis of 40 per cent, 24 per cent of the tonnage broken is required for packing and it is necessary to break 1.32 times the output tonnage; with 60 per cent coverage the broken tonnage becomes 1.56 times the output tonnage. The percentage of area covered depends upon the top pressure; in mines where the top pressure is great, larger percentages are covered. The open area between the face and the pack depends upon the roof conditions; it ranges from 6 to 12 ft. Tim-

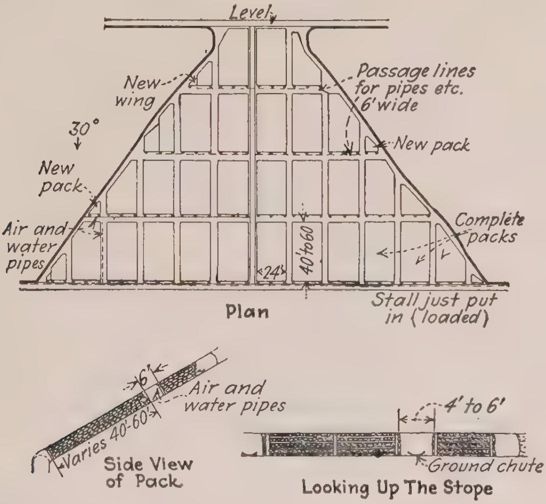
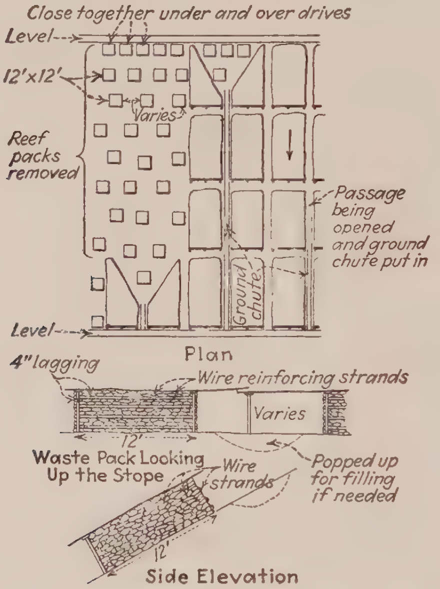


FIG. 258.—South African ore-packing system of support. (*Paul Selby.*)

bers are used and these are retrieved as far as possible as the pack advances. When the packs are kept well up to the face, 3 to 4 ft., very little shoveling is required. A face 200 ft. in length advances three-quarters of a foot per shift, so that the pack wall is extended about 18 in. per day. In heavy ground, packs are reinforced with strands of used wire rope, cut in lengths of 14 to 16 ft. and placed across and along the length of the pack, the bent ends being carried around the lagging of the stull or big stones in the walls of the pack. Generally packs are left until all the stoping upon a level has been finished, but this practice is not always followed. Assuming that a sufficient area has been mined, reclaiming of the remaining broken ore takes place. During this period the rate of reclaiming is determined by the tonnage demand. As the hanging wall settles gradually upon the packs and does not break up in detail, the ore is reclaimed without difficulty.


Close together under and over drives
Level
12'x12'
Varies
Reef packs removed
Passage being opened and ground chute put in
Ground chute
Level
4" lagging
Wire reinforcing strands
Varies
12'
Wire strands
Popped up for filling if needed
12'

Plan

Waste Pack Looking Up the Stope

Side Elevation

Fig. 259.—Reclaiming of broken ore packs. (*Paul Selby.*)

Figure 259 shows the plan of a partially worked-out stope. A footwall chute is laid between two rows of packs. Workers start at the topmost row of packs and move sufficient ore into the chute to permit the construction of a waste-rock pack. As the packs are moved all big pieces are washed, and the waste is placed to one side. Any hanging-wall or footwall bands left after the first stoping are taken out if payable and the footwall is scraped and washed. Waste-rock packs are built upon the clean footwall where required, sizes ranging from 8 by 8 ft. to 8 by

12 ft. and 12 by 12 ft. They are built on stulls, 8- or 12-ft. lagging being supported by two posts. Insufficient waste rock is compensated for by drilling and blasting pop holes in the footwall between the waste packs. Reinforcing with wire rope strands is put in where greater strength is required. Waste-rock packs are placed at intervals of 8 ft. or more, the condition of the hanging wall determining this interval. They are, however, placed closer together along traveling ways in the stopes and along the top and bottom sides of the main drifts. Generally two rows of packs are removed, one on each side of the chute lines. As the work of reclamation approaches the lower level, another chute line is established. Chutes are shallow curved metal troughs. Water is used in facilitating the downward movement of the ore.

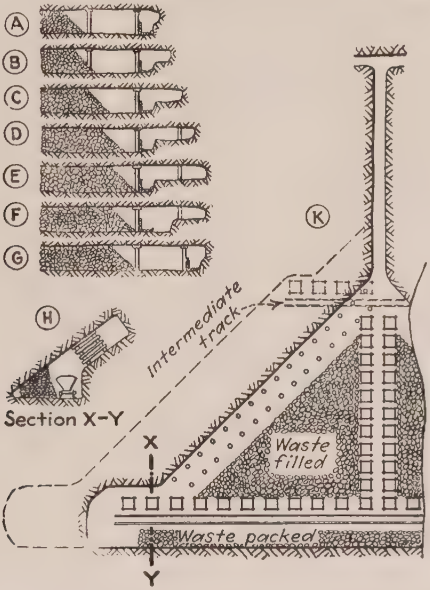


Fig. 260.—Resuing at the City Deep Mine. (*A. V. Lange.*)

A. V. Lange states that in mining at depths below 5,000 ft. (referring to South African conditions) support of the hanging wall becomes one of the most important and most costly operations, not only on account of safety at the working faces but also to provide good general support so as to reduce concentrated pressure in newer and lower workings. In working places stoped at less than 48 in. width, reclamation of ore left behind becomes most difficult and expensive if it is not carried out within 60 ft. of the face. Owing to increased rock temperatures, more air is required for ventilation of the stope faces.

At the City Deep, Ltd., a resuing system of mining is applied at great depths. The details of the system, as described by Lange, are as follows:

The ore is overcut by stoping waste rock or unpayable hanging-wall bands, the advance being about 8 to 10 ft. The waste rock is packed in the finished working behind. The breaking of this overcut is between partings of the formation. Upon its completion the ore upon the footwall is broken out and removed by mechanical scrapers to the lower level. The sequence of the steps is shown in Fig. 260, which also shows the plan of the working. It should be noted that before resuing starts, a row of 7-in. props, 7 ft. apart, is placed at a distance of 2 ft. from the face. Lagging boards are placed on the inside up to the height of the waste wall, and a line of chutes is placed along the footwall between the lagging barricade and the face. The angle of the face is selected so as to secure a dip of 26 deg. along the face to facilitate ore movement during the ore-mining period. Where mechanical scrapers are used the face can be worked at a flatter angle. Footwalling of ore is done by pop-holing and by barring with heavy machines weighing 50 to 65 lb. and designed as concrete breakers. Footwalling is carried to within 1 or 2 ft. from the waste-rock face so as to present a ledge for the next attack. The length of face worked as a unit, 150 to 180 ft., is maintained as straight as possible.

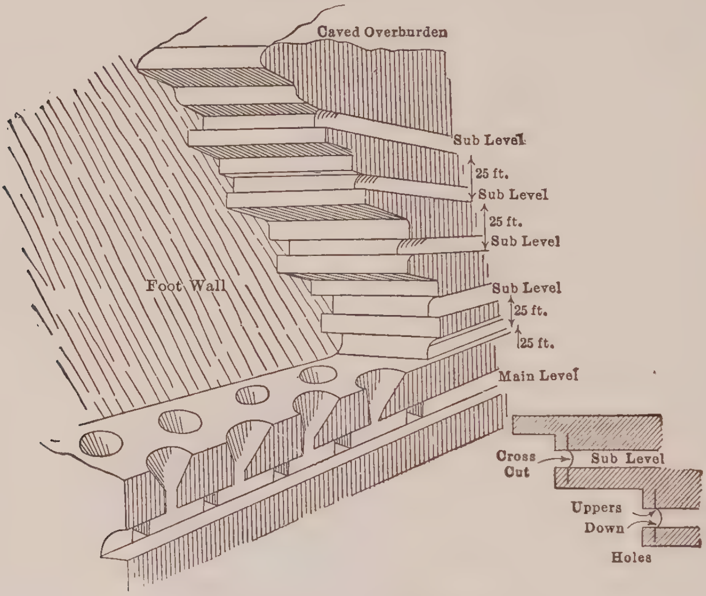


FIG. 261.—Sublevel stoping method.

The Sublevel Method.—This method, used in the Michigan iron mines, is applied to steeply dipping ore bodies in which the ore is of such strength that it will stand unsupported across the width of the stope. It can be considered as a modification of overhand stoping. Figure 261 gives a general idea of the method. A main level is opened out, and sublevels connected by raises are driven about it at intervals of 25 ft. The first sublevel is opened out as a chute level, and funnel-shaped chutes that connect with the main level are opened out from it. These are spaced at the 25-ft. intervals, center to center. The first sublevel is excavated to a height of 16 ft. and is extended by the driving of a crosscut 8 ft. high from the end of the sublevel to either wall. Holes 8 ft. deep are placed in the back and a slice 8 ft. thick is blasted down. This process is repeated in retreating along the sublevel. As soon as operations on this sublevel have prepared chute openings, work begins on the horizontal slice above. A crosscut 8 ft. high is driven from the end of the sublevel, and from this down holes are drilled in the 8-ft. bench beneath and uppers are drilled in the bench above. These are blasted, the broken ore falling into the open chamber beneath, from which it is drawn through the chutes. A second crosscut is driven, and the drilling of the upper and lower bench from this crosscut follows. The sublevels above are started sequentially, and each follows the one beneath in a retreat from one end of the body to the other. The overlying rock falls into the stope on top of the broken ore. The obvious advantage of the method over shrinkage stoping is the better protection of the miners, since they work under a solid back at all times and never in the open stope. The stope need not be kept filled but can be drawn off as rapidly as circumstances require.

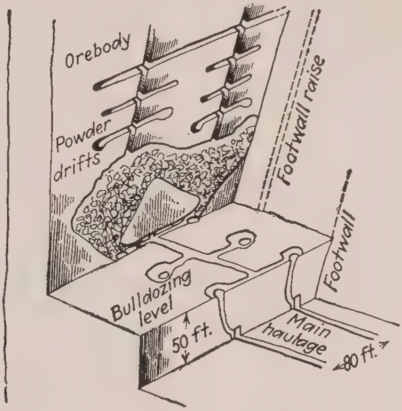


Fig. 262.—Alaska Juneau system of mining. (*P. R. Bradley.*)

Several interesting methods of sublevel stoping have been applied to large low-grade ore bodies. At the Alaska Juneau property a grizzly or bulldozing level is established above a main haulage level and connected to it by chute transfers. A three-bar grizzly with 25-in. spacing between bars closes the top of each transfer. Above the grizzly a 38-deg. raise extends to the limits of the stoping area. From each pair of grizzly raises on each side of the grizzly level, drift raises are taken off a short distance above the grizzly and extended in the opposite direction until they meet. From these raises the bottom of the stope is cut out along its length, pillars being left for support. In this condition the stope bottom has a section resembling the letter W (Fig. 262). Stope raises are then extended paralleling the dip at 100-ft. intervals along the length of the stope and rising from every other pair of grizzly raises, which are spaced approximately 40 ft. apart along the length of the stope. Sublevels (4 by 3 ft.) at vertical intervals of 35 ft. are opened out from the stope raises and fan out in different directions. The stope raises connect with an upper level. The sublevels are used as "powder" drifts, and stoping begins upon the completion of the cutting out and raising. The lowest sublevels are first blasted, each being charged with about 4,000 lb. of 40 per cent dynamite distributed in two piles, 35 or 40 ft. apart. The average burden on each blast is about 35 ft. Cordeau-Bickford detonating fuse is laid from pile to pile. The detonating fuse is detonated by No. 8 blasting caps and two 30-ft. lengths of single-tape safety fuse. Explosives are taken to the sublevels through the raise from the upper level. Each pile of explosive is tamped with about 15 ft. of fine-rock stemming. About 20 tons of ore is broken per pound of explosive. This high ratio is partly due to caving. The broken ore is drawn off to the grizzlies and oversize is blockholed and blasted. Figure 263 gives details of the bulldozing chamber. A considerable proportion of oversize results from this method of blasting and caving. P. R. Bradley states that of the total consumption of explosives, 13 per cent is used in development and in preparation of stopes, 16 per cent in stoping, and 71 per cent in secondary blasting or blockholing.

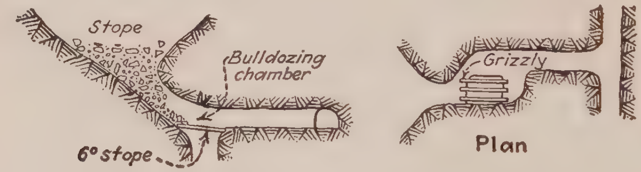


Fig. 263.—Alaska Juneau bulldozing chamber. The grizzly bars are 16 ft. in length, 15-in. 104 H beam, with 13-in. channels riveted to tops; cast-iron stiffeners are riveted at 4-ft. intervals on each center grizzly bar and on the outside of the side bars to stiffen the wide flanges. (*P. R. Bradley.*)

Subsequent to the foregoing practice, drilling chambers were opened out in the raises, and from these blastholes were fanned out. This avoided the construction of the sublevels and gave greater control over blasting operations.

In the Latouche system of sublevel stoping, the ore body is divided into alternate stopes and pillars, the stope width being 70 ft. and the pillar 30 ft., both being transverse to the strike of the ore body. A haulage level is established about 50 ft. above a grizzly level, from which short transfers extend to the stopes. At a distance of 150 ft. above the grizzly level the principal mining level is established. From this, access is obtained to the system of raises by which the ore is worked. A hundred feet above the principal mining level is another mining level giving access to higher portions of the ore body. Each stope width is covered by two series of raises at the angle of dip. The stopes are undercut and connected to the transfers to the grizzly level by funnel-shaped excavations. There are two lines of grizzly chambers 15 ft. from the pillar limit to serve each stope and one line to serve each pillar. The grizzly chambers are 24 ft. by 8 ft. and 7½ ft. in height and are spaced from 60 to 75 ft. center to center from the footwall to the hanging wall of the ore body. The mining raises are 5 by 5 ft. in section. Figure 264 shows a section of a stope after it has been in operation. Figure 265 is a sectional view of the grizzly chamber.

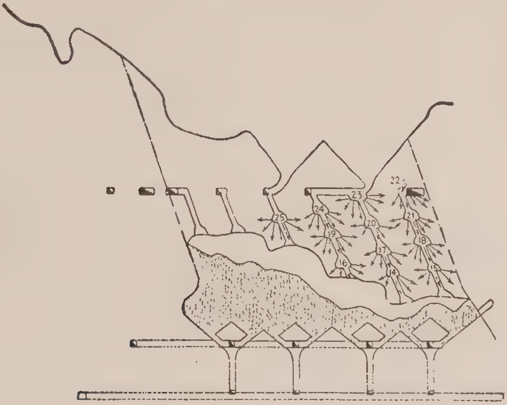


Fig. 264.—Longitudinal section of stope after 4 months' mining. Numbers show order of blasting rounds. Latouche system of mining. (*Bevan Presley.*)

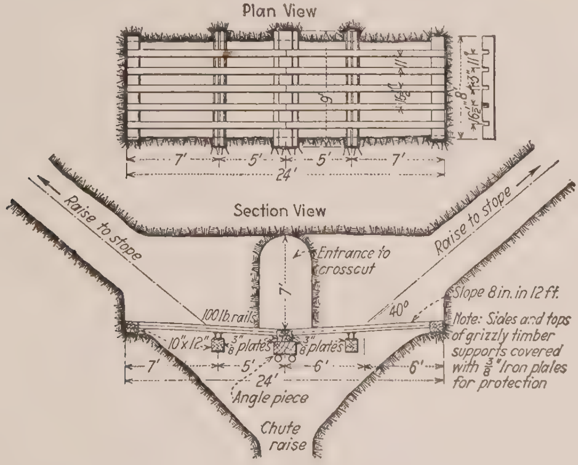


Fig. 265.—Latouche grizzly chamber; 100-lb. rails, spaced 16½ in. centers. (*Bevan Presley.*)

Mining is started in the footwall raises. At a point 30 to 40 ft. above the back of the undercut stope an enlargement is made in each raise, forming benches 18 to 25 ft. in length and 12 to 14 ft. wide. The floor of the bench about the raise is 20 to 30 ft. above the back of the stope. A series of deep drill holes are drilled from the benches and fan out so as to command a stope area of about 52 by 35 ft., the longer dimension being parallel to the pillar. In hard ground 35 to 40 holes are drilled, and in softer material 20 to 26. Drills up to 24 ft. in length are used, 2⅛-in. starters and 1¼-in. finishing diameter. After a round is drilled the holes are sprung from 1 to 3 times and are thoroughly cleaned out by means of an air blowpipe. The round is charged with 40 per cent explosive; No. 8 blasting caps are used in parallel series and from 6 to 8 delays, depending upon the number of holes in a round. A large blasting machine or 110-volt lighting circuit is used for blasting. The amount of explosive varies from 500 to 1,000 lb. per round, giving an explosive consumption of 0.245 lb. per ton. A complete cycle requires from 14 to 17 days and from 28 to 34 man-shifts. Successive drilling benches are opened out in the mining raises at higher and higher points and from footwall to hanging wall, as is indicated in the figure. The pillars are mined in a similar manner.

The foregoing examples differ from the usual method of sublevel stoping in that the retreat is in an upward direction instead of horizontally from one end of the stope to the other. In both cases a very low mining cost was obtained.

3. Combined Underhand and Overhand Stoping.—In veins of moderate dip, 45 to 50 deg., it is sometimes convenient to combine overhand and underhand stoping, as this method gives more working places in starting a stope.

4. Square-set Stoping.—Where square-set timbering is used to support the ground as the ore is taken out, the method of mining is called the "square-set system." This method is most often used for winning wide ore bodies; but it finds peculiar application where the ore body is irregular either on account of indefinite walls, offshoots of ore as in a system of linked veins, or where faulting has divided the ore body; where much waste or horses of barren rock are features of the ore body; where the ore changes its grade rapidly and close sampling is required; where walls and ore bodies are weak; where low dip and a heavy hanging wall prevent shrinkage stoping; where walls are moderately strong, the ore body is weak, and top slicing cannot be used; where high temperatures prevail; and where the ore body is too wide for stull timbering or for the ordinary waste-filled or shrinkage stope. In top slicing, square sets are usually used for the support of the rooms upon the uppermost slice. The irregularity of the top of the ore body can be better followed than with the ordinary drift sets. In some applications of top slicing the square set or a modified set closely resembling it is used upon all the slices. It has the advantage of allowing a somewhat thicker slice than can be taken with the system usually followed. Four general type divisions are made: Open stopes, partially filled stopes, filled stopes, and alternate pillar and stope. In all applications of square-set stoping the ore is excavated in blocks of approximately the same size, ranging from 5 by 5 by 7 ft. to 6 by 6 by 8 ft., the largest dimension being the vertical. As soon as one or more blocks have been removed, a timber frame (square set) is erected in the open space. The members of a set are arranged to fit into the sets already in place. The details of the square set are given in the chapter on Support. Blocking or braces support the timbers next to the ore and wall. Where walls and ore back are strong, a number of blocks may be excavated before the square sets are placed; but where the ground is heavy, one block at a time is removed and timbered.

The stope is started from a raise that is timbered with square sets. A horizontal slice, one set high and extending over the level from wall to wall, is removed and timbered. The first horizontal slice is called the sill floor. The posts of the sets rest upon sills, which are usually placed transversely to the walls and which are long enough to carry posts of two or three sets (10 to 15 ft.). When the first slice has been extended, another slice immediately above, called the first floor, is started. The floors above are started in sequence as soon as the lower floor has been advanced sufficiently to prevent the operations upon it from interfering with the work on the lower floor. Heavy plank floors are laid upon the timbers and afford working platforms for the miners. The broken ore falls upon the platforms and is removed to ore passes, which are formed by planking the sides of a vertical series of sets. The chutes or passes are placed at from 25- to 30-ft. centers in both directions and terminate upon the sill floor, at which point the chute gates are placed and loading into cars is effected. When chutes are closely spaced, the broken ore is shoveled directly into the chute. With 50-ft. spacing, a single line of chutes, wheelbarrows, or short inclined chutes are used to convey the ore to the chute. In breaking the ore, uppers from the lower side (back) of the block or down holes from the side may be used. Access to the stope is afforded by ladderways or stairways. Ventilation is effected through the raise which connects the levels and through which timber may be lowered and compressed-air pipes are extended. The stope is extended upward until it joins the square sets of the level above. The long sill pieces enable the connection to be made with a fair degree of facility and safety. Careful work entailing light blasting is required in making the connection.

Open Stopes.—The square-set system was designed originally to maintain the stope as an open-timbered chamber. Deidesheimer (its inventor) employed it successfully in mining ore bodies up to 150 ft. wide and several hundred feet high and long. He employed very heavy

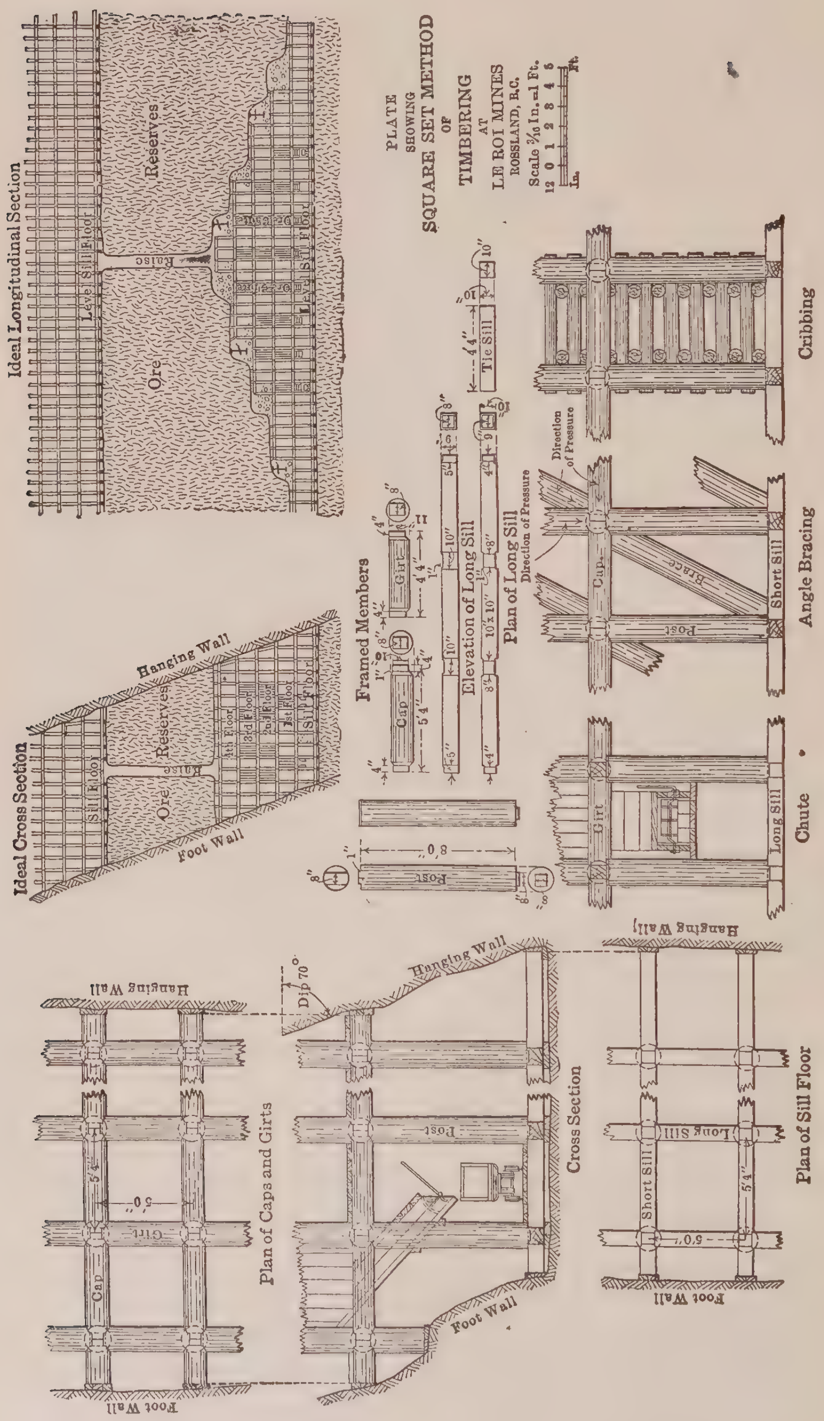


FIG. 266.—Square-set stoping. (*C.I.M. & M.*)

timbers, 14 by 14 in., and in addition used wall plates and heavy diagonal braces. In modern applications of the system lighter timbers are used (12 by 12 and 10 by 10 in.); wall plates are used only where lagging is required on the hanging wall, while diagonals are generally omitted or used only when the sets begin to lose their alignment. As might be expected, heavy ground and wide ore bodies give trouble with such modifications. Evidence of trouble is given by the sets' losing their alignment. The integrity of the system depends upon the alignment of all the members, and the resistance of the timbers to top and side pressure is materially lessened when they begin to move. Diagonals and reinforcing sets are placed to save a stope when this condition becomes apparent. Reinforcing sets may be placed under caps, or girts, while the diagonals are placed transverse to the walls. Local weakening of the timbers of an open square-set stope is remedied in the same manner. If these expedients do not suffice, cribbing is placed within tiers of sets. Filling is the only remedy where cribbing fails to hold the sets in place. In present mining practice the filled stope has superseded the open stope, the latter being used only for small ore bodies. Figure 266 illustrates the system.

Partially Filled Stopes.—Under some conditions it may be desirable to resort to partial filling instead of the practically complete filling usually employed. Where this is done, a portion of the sets, four or five along the strike and extending from wall to wall, are laced or cribbed on the ends and the space between is filled with waste rock. The result is a rib of waste rock reinforced with timber and extending across the ore body from wall to wall. Several such ribs placed along the length of a stope have the effect of steadying the whole structure.

Filled Square-set Stopes.—Three methods are in use, the ordinary back stope and waste fill, the incline cut and fill, and the vertical slice and fill. As will be seen from the descriptions, there is more or less similarity between the methods.

Back-stoping and Waste Filling.—The sill floor is worked out over the area of the block and square-set timbers are placed. The first floor is then started, extending from a two-compartment winze that connects with the level above. One compartment of the winze is boarded and serves as a waste chute. While the first floor is being extended, waste is loaded into cars from this chute, trammed over temporary tracks, and dumped into the open-sill sets. Levels and crosscuts where required are protected on top and sides by heavy lagging. By the time the first floor is finished, the sill sets are filled and another floor is started. In effect, two floors are always open where the miners are working.

Incline Cut and Fill.—The stope is started at a winze and waste chute. Instead of carrying horizontal slices the stope face is stepped, the edges of the steps falling upon a 45-deg. plane. After a diagonal line of square sets is placed, filling is run until it stands at the angle of repose. Planking is then set in place, forming a run to the nearest chute, and the ore is broken down and worked down to the chute. Both fill and stope face maintain approximate parallelism. The stope is worked intermittently, first excavation and timbering, then waste filling, then planking, then ore breaking and timbering, followed by the removal of the planking, and so on. Square sets above the sill floor are best made of the same vertical height as the center-to-center horizontal dimensions, for example 6 by 6 by 6 ft. This arrangement will permit the planks, which prevent ore and waste from mixing, to be placed upon the fill. The main advantage of this method is that little or no shoveling is required, as both the ore and waste are handled by inclined chutes.

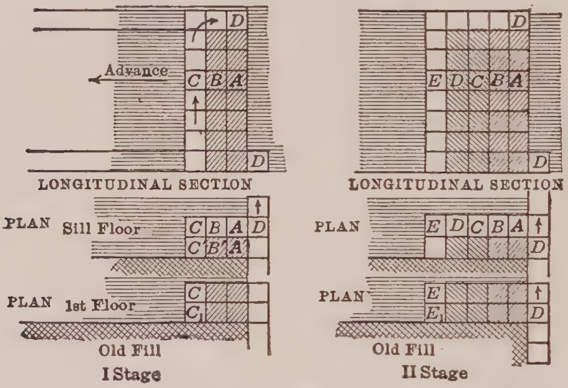


FIG. 267.—Vertical-slice method of square setting.

Vertical Slice and Fill.—Where this method is applied the block height is usually limited to five or six sets above the sill floor. The slice, which may be one, two, or three sets wide, extends from wall to wall. It is handled as if it were an ore body of limited length and moderate thickness. The ore is removed by back-stoping or incline cut, and the stope is timbered with square sets. When the slice has been completely removed, the side next the standing wall is laced with 2-in. plank, and the stope is filled with waste from the level above. ("Lacing" is a miner's term and signifies the nailing of wooden strips between which open spaces 2 to 4 in. wide are left.) In some cases the sets adjoining the ore block are left open and the others filled. This leaves the new face accessible on the entire side. The narrow stopes can be laid out parallel with the length of the ore body, but the former method is usually followed. Where the ore body is too heavy to admit of maintaining the narrow stope as an open stope, it may be removed in vertical open blocks of a bottom area of four sets. The ore is removed and the space timbered. The sets are then filled from the top, the vertical line of sets in the direction of the advance being left open. Figure 267 illustrates the method. *D* represents the top and bottom drifts, which are left open for air and the handling of waste and ore. One horizontal line of sill floor sets is left open for the same purpose. The procedure is to open out sets *A*, *B*, *C*, *A′*, *B′*, *C′*. The side of *B* next to *C* is laced, and sets *A*, *B*, *A′*, and *B′* are filled, *C* and *C′* remaining open. *D*, *E*, *D′*, and *E′* are then taken out and *C*, *C′*, *D*, and *D′* are filled. A modification of the method is to stope down a pair of sets from the top to the sill-floor level. The stope starts from a raise and is never more than two sets wide and one set in advance. The square sets are put in from the top down, being supported at the top by nailing planks, booms, and diagonal braces. Figure 268 illustrates the method of supporting the square sets. On account of the weight of the sets, the vertical height is limited to four or five sets.

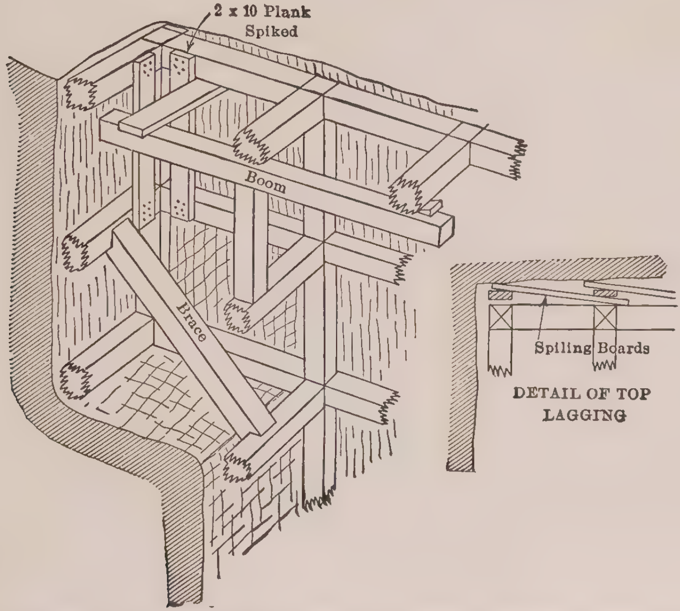


FIG. 268.—Square setting from top downwards and using booms and braces for supporting timbers.

Alternate Pillar and Stope.—Where the ore body is of considerable length it is divided into sections, and every alternate section is worked as an independent stope. It is evident that any of the foregoing modifications could be used in the section. The physical nature of the ores and walls would determine the most advantageous method. It is not unusual to plank the sides of the filled stopes; the pillar is then ready to be extracted. Transverse or longitudinal narrow stopes or slices could be used in mining the pillar. Wide and long ore bodies are divided into blocks, and these blocks are mined just as if each were a separate ore body. The stope is filled as the block is mined. Two or four contiguous blocks may often be mined simultaneously as shown in Fig. 269, which represents the practice at the Copper Queen mine.

One of the best examples of the application of square set and waste-rock fill is afforded by the Sullivan mine of the Bunker Hill and Sullivan Mining Co. Ore bodies in this mine vary from 300 to 1,000 ft. in length and from 30 to 125 ft. in width, greatly brecciated and with numerous small faults and slips; the hanging wall is very heavy and cannot be supported by timbers alone. The initial attack is made at the hanging wall, and the waste-rock fill is started as the stope is opened out. The waste-rock pack supports the hanging wall, and the footwall supports the ore in place. A system of vertical slices is practiced, the advance from the raise sets being in both directions along the strike and toward the footwall, so that the stope maintains a more or less arched section in both directions, *i.e.*, in the vertical longitudinal and transverse planes. Figure 270 shows the successive stages in stoping. A drift is started in the footwall and extends to the ends of the ore body, crosscuts being extended to the hanging. At the central point of the ore shoot a raise is driven in the footwall to the level above and serves as a waste pass, timber pass, and manway; chutes are established at the topmost floor of the square sets so that the waste rock can be trammed in at the highest point of the stope, which is started at one of the crosscuts intersecting the hanging wall. The square-set timbering of the crosscut is extended upward as the stope rises. After a stope is started, a drift is extended in the footwall at a distance of 20 to 30 ft. from the ore body to provide a permanent haulageway. Raises are then extended from the drift approximately parallel to the footwall, and branch raises are extended from these into the stope. The footwall raise is not timbered, but the

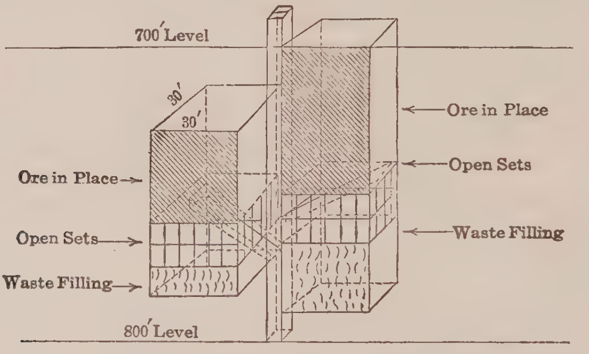


FIG. 269.—Alternate pillar-and-stope method, Copper Queen mine. (*Trans. A.I.M.E.*)

branches are usually cribbed. The footwall raises and their branches serve as ore transfers. Waste rock is secured from development work and also by driving raises in soft ground away from the stoping area, where natural caving will maintain a supply of waste rock.

The mining of large ore bodies has brought about a variety of systems, most of which are combinations of features found in the simple methods. At the Calumet and Arizona Junction mine the Campbell ore body is mined by a modification of the incline cut and fill. The ore body is divided into sections on a given level, and each alternate section or block is stoped as a unit. Parallel crosscuts are driven the length of the block and timbered by posts and stringers, as shown in Fig. 271. Narrow cuts are made on each side of the block and timbered with square sets. The floor of the stope is opened out from the first tier of lead sets, the cut being made on an angle of some 37 deg. from both sides, forming a ridge. This is covered with stringers and waste lagging so as to form a mat. In the meantime a raise has been extended to the level above, which serves as a waste-rock transfer and for ventilation after its timbers are stripped. Lead sets on both sides are then lagged, and waste-rock fill is introduced up to 2 or 3 ft. from the back. The waste-rock fill is then covered with 2- by 10-in. plank laid on 2- by 10-in. sills, care being taken to avoid hollow places between the planking and the fill. Grizzlies of 10- by 10-in. timbers are then placed in the lead sets at the intersection of the fill floor line and the lead sets. A stope cut is started at the bottom, the average height of cut being 12 ft. Sufficient broken ore is left to give a working position, and in bad ground "umbrella" stulls are placed to

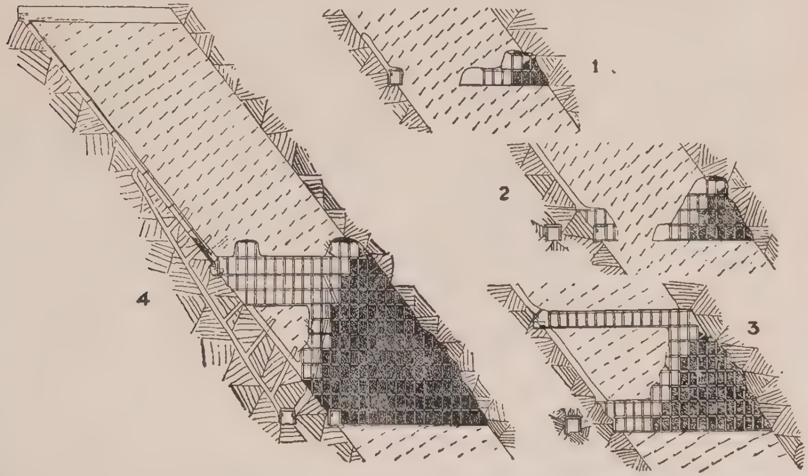


Fig. 270.—Method of vertical slicing, square-set timbering, and waste-rock filling employed at the Sullivan mine. (*J. W. Gwinn and W. E. Brown.*)

support the bad area. After the cut is completed from both sides the broken ore is drawn into the chutes, the sloping floors are thoroughly cleaned, and the planking is removed. The sides of the lead sets are then laced with lagging and a new layer of waste rock is placed. Each routine of cut and fill is the same, and the lead sets are raised along with the stope cut.

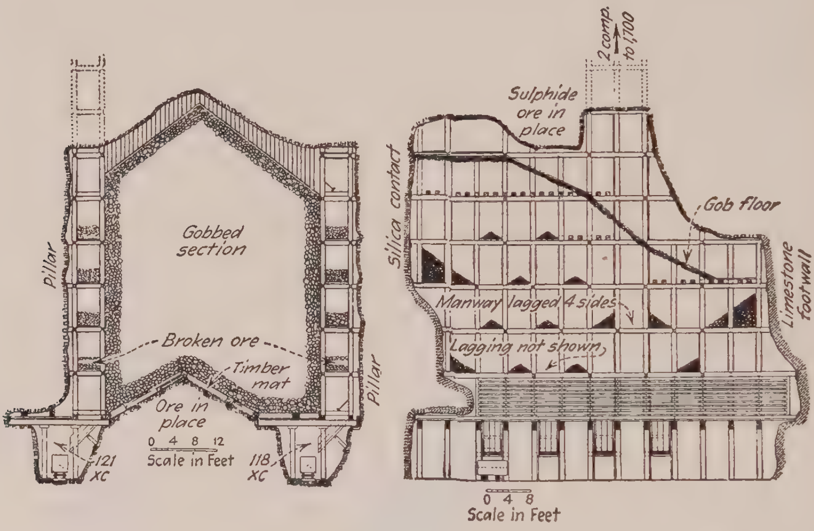


Fig. 271.—Incline cut-and-fill stope with lead stopes. (*H. M. Lavender.*)

A modification of the foregoing is employed where the back is uniformly good. In this modification, successive cuts are taken after the completion of the stope flooring, the surplus broken ore being drawn off through the lead sets at the sides. When the successive cuts have reached a height limited to about 40 ft. above the sill floor, all the broken ore is removed, waste-rock filling is introduced up to 2 or 3 ft. from the back, and flooring planks are laid as in the first example. A new ore cut is then started. In this manner the frequent placing and removing of flooring are avoided and both labor and timber are reduced in amount.

5. Shrinkage Stoping.—The principle of shrinkage stoping has already been given. In mining large ore bodies it can be applied where the ore body itself is self-sustaining across its width and where the walls are sufficiently firm to stand without support over a considerable length along the strike of the vein. The range of dip permissible is from 50 to 90 deg. Overhand stoping is almost invariably applied, although some instances of rill stoping occur. A pillar of ore is usually left above the lower level, and the ore chutes are cut through this to the lowest slice. The first slice is driven through and the mouths of the chutes are widened so as to form funnel-like openings. Sufficient ore is removed to leave working room. Large pieces of ore are blockholed and blasted. The second slice is then started, the broken ore serving as a working platform. The back is arched so as to make it more self-supporting. The stope is continued up in this manner until close to the level above, below which a pillar is left to protect the top of the stope and the level above. The broken ore left in the stope can then be drawn off. The ends of the stope in an ore body of considerable length are protected by vertical ribs of ore, through which extend the manways giving access to the stope. At different points short drifts through the rib connect the manways with the stope. The recovery of the top pillar can be effected by filling the stope with waste and then square-setting the pillar. The end pillars can be won by square-setting also.

A modification of the ordinary system of shrinkage stoping is occasionally met with. The principal difference is that ore passes are carried up at intervals along the line of the stope and surplus ore is removed through them. This leaves the broken ore in the stope in stable condition. Temporary supports can be constructed on the top of the broken ore, so that a back too weak for the usual shrinkage stoping may be supported sufficiently to make the working safe.

The method is not confined to wide veins alone but may be applied to narrow, steeply inclined veins. With veins of great length it is customary to separate the stopes by pillars. In the Homestake mine the stopes are 60 ft. wide along the length of the vein and extend from hanging wall to footwall. Pillars 42 ft. wide separate the stopes. In the Alaska Treadwell the stopes were lengthwise with the strike and were 200 to 300 ft. long and from 160 to 250 ft. wide. The vertical height of a stope depends upon the dip of the vein. With flat veins of moderate thickness and 50 to 55 deg. dip, a level interval of 100 ft. is used. For a vein of 60 deg. dip, a level interval of 150 to 175 ft. is permissible. At the Melones mine two levels were 285 ft. apart. It is evident that an increased interval between levels prolongs the time required to complete a stope, and until the stope has been completed only the surplus ore can be drawn off. This surplus ordinarily amounts to one-third of the ore broken, so that for every ton of ore produced from a given stope 3 tons must be broken and 2 tons left in the stope. It is economy to use as great a distance between levels as possible, both on account of the reduction in the cost of levels and the smaller loss of ore in pillars. For ore bodies of moderate length and height it is possible to plan for the winning of the ore body in one stope from a single level. Figure 272 is typical of shrinkage stoping.

Where narrow and comparatively rich ore bodies are mined by shrinkage stoping, the cleaning down of the empty stope is important. It can be done by the miners entering the stope. As the broken ore is drawn off, the walls are cleaned down and are timbered with stulls, where necessary, to make them safe. The work of cleaning and timbering follows the drawing off of the ore. Shrinkage stopes may be left open or filled with waste. Filling may be placed simultaneously with the drawing of the broken ore by drawing one end of the stope out completely and then beginning the fill at this end. The line of complete drawing off is followed by the line of the waste fill. The names suggested by these modifications of shrinkage stoping are "longitudinal shrinkage stoping," "transverse shrinkage stoping," "shrinkage stoping and filling," and "shrinkage stoping with backfill."

The obvious advantages of shrinkage stoping require no comment. The disadvantages are as follows: All the vein material must be mined and removed from the stope, whether it be waste or ore; it takes time to extend a stope from level to level, and while this is being done a large tonnage must be broken and must remain in the stopes for a considerable period; the grade of ore coming from the stope may fluctuate considerably; the flattening of the ore body between levels may greatly interfere with the drawing off of the broken ore; more or less ore is left in pillars and in stopes. One essential condition must not be overlooked: The physical nature of the ore should be such as to permit it to be readily drawn off when broken. An ore that packs tightly when broken cannot be mined by this method. It should be noted further than an ore that would pack would also be apt to be too heavy to admit of the use of the system. In mining certain copper ores by shrinkage stoping, oxidation of the ore takes place during the interval in which the stope remains filled. In sulphide ores this oxidation interferes with concentration, and therefore the method cannot be used.

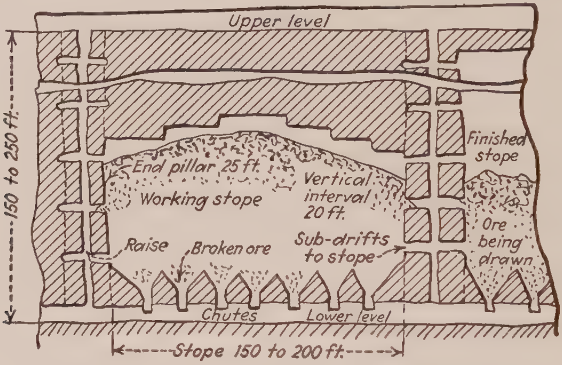


FIG. 272.—Longitudinal section of shrinkage stope.

The following description of shrinkage-stoping practice at the Homestake mine is abstracted from an article by A. J. M. Ross. Levels are 150 ft. apart. When the ore body is encountered on a level, exploratory drifts are driven in the ore parallel to the walls, and the shape of the ore body is determined. Crosscuts are then driven from these drifts in the centers of the pillars and into both hanging walls and footwalls. The crosscuts are connected by drifts in both walls at an approximate distance of 25 ft. from the ore body. The ore body is divided into 42-ft. pillars and 60-ft. stopes. Manway raises are cut from level to level on the footwall, and short sublevels are extended at a number of points to the stope limits, as shown in Fig. 273. These give access to the stopes when the mining operations intersect them. Figure 273*b* shows the plan of the stope at the level. The entire bottom area of the stope is mined out, and two lines of chutes are erected next to the pillar walls.

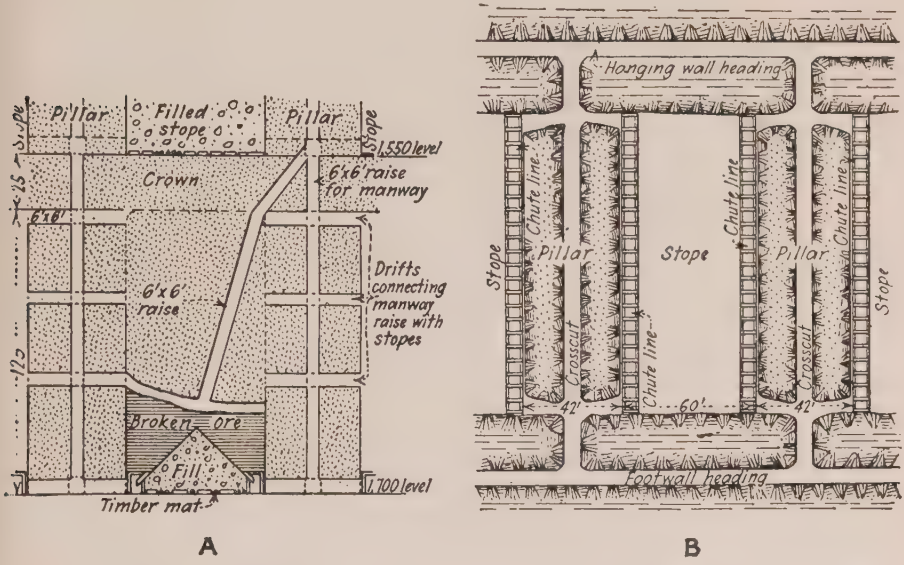


FIG. 273.—Longitudinal section of stope and plan. Homestake mine. (*A. J. M. Ross.*)

The sill floor is covered with a timber mat. Waste rock is run in from transfers, the bottom fill being sloped from the center line to the chute lines on either side. Shrinkage stoping then begins and the stope is extended upward within 25 ft. of the level above. The longitudinal rib is won by square-setting, as is shown in Fig. 274. The pillars are removed by overhand stoping with square-set timbering in two blocks each 75 ft. in height, as shown in Fig. 275. Some shrinkage stopes have been carried up to a height of 275 ft. with no loss of ore and without dilution from wall rocks. In narrower portions of the ore body, shrinkage stopes are carried parallel with the length of the ore body and are 100 ft. in length, separated by pillars 25 ft. in length, in which the manways and sublevels giving access to the stope are cut.

6. Top Slicing and Cover Caving.—Ore bodies of considerable lateral extent and of small or considerable vertical range are mined by this method where they occur under a noncoherent or weakly coherent covering and the ore itself is soft. The method is a retreating one; the ore body is worked from the top downward in horizontal slices, and each slice is worked inward from the boundaries. The thickness of the slices varies from a minimum of 7 to 9 ft. to a maximum of 12 or 16 ft. The nature of the ore and the methods of support determine the thickness

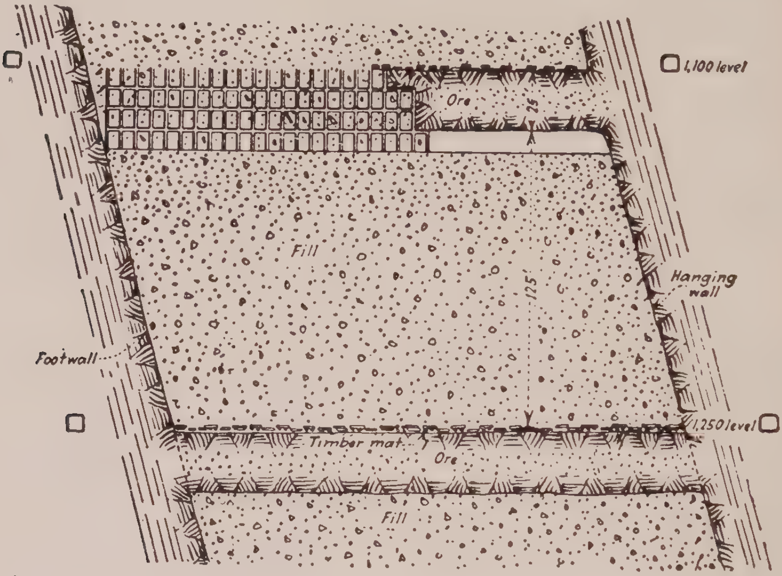


Fig. 274.—Mining of rib below level. Homestake mine. (*A. J. M. Ross.*)

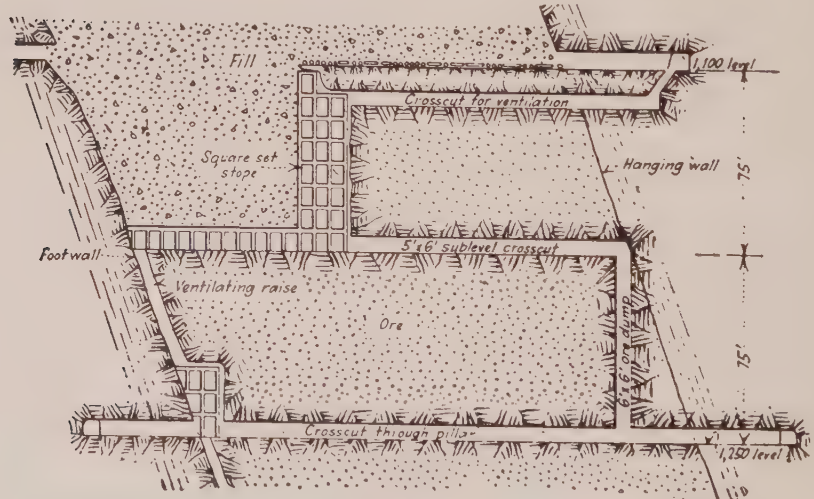


Fig. 275.—Method of mining pillars between stopes. Homestake mine. (*A. J. M. Ross.*)

of the slice. Economical working requires as thick a slice as possible. Working shafts are placed outside the area of the deposit. A main haulage level is extended from the shaft through the lowest part of the deposit or at an intermediate point if the vertical range is considerable. Sublevels connected to raises and chutes terminating at the main haulage level provide access and ventilation. Several slices may be worked simultaneously, the upper slice in every instance being worked 50 to 100 ft. in advance of the one immediately below (see Fig. 276).

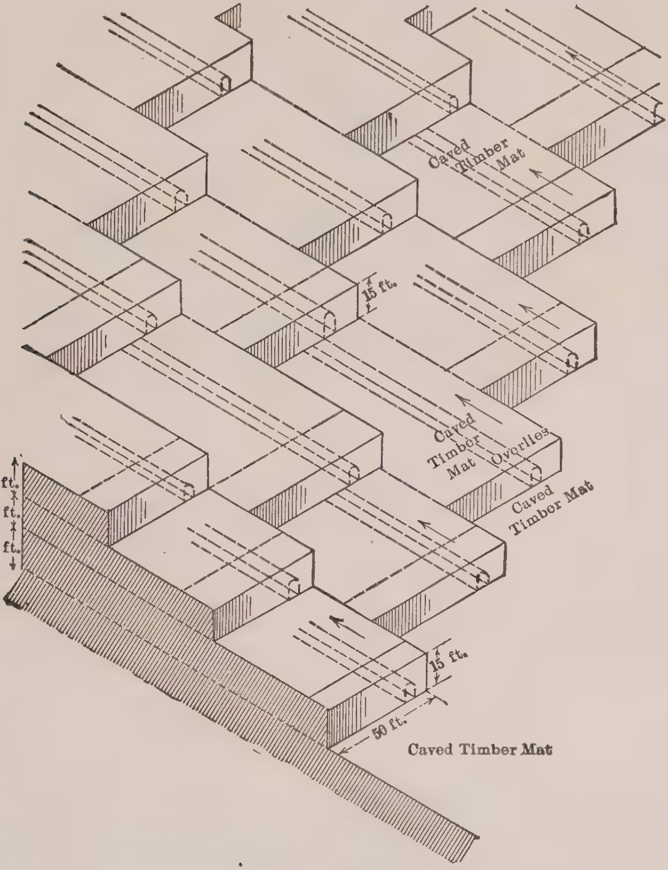


Fig. 276.—Top slicing.

Figure 277 shows the sequence of mining a top-slice room in iron ore. The crosscut interval and the room length are both 50 ft. Support is by two rows of drift or three-piece sets spaced at 5 ft. down the length of the room. The initial break is toward the cave side and at the top of the section. Lagging boards are placed upon the leading cap, with the other ends resting on the upper edge of the ore face. The block is mined down to the floor; the ore is then removed and a new set is erected in the space close to the face. Hand loading has been supplanted by scraper loading and mechanical shovels. When a room has been completed, a floor of 1-in. plank is laid down and the sides of the room and its entrance are boarded. The center posts are drilled and blasted simultaneously, allowing the top to cave, which may take more or less time. Another unit alongside may be started as soon as caving takes place. The caved timbers eventually form a mat of considerable thickness, which gives better control of the caving and also is a factor in preventing dilution by the overburden. Ventilation during the mining of a room is by Ventube and blower.

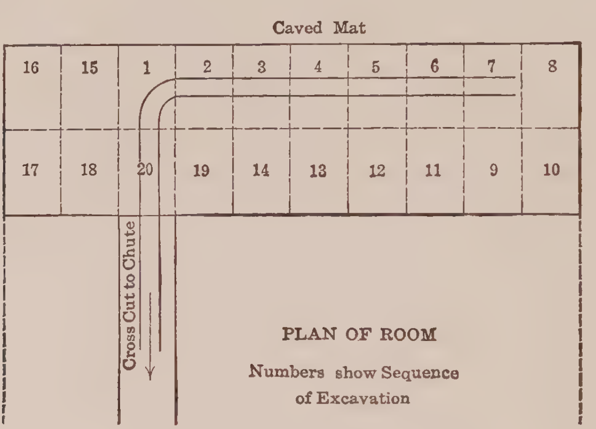


Fig. 277.—Sequence of mining a top-slice unit.

Moderately hard overburden permits greater unit areas to be mined. At the Mammoth mine (California) the unit was 40 × 40 × 9 ft. high. Square sets, posts and headboards, and posts and stringers have been used for top support. At Cananea round props were used; when a room was ready to be abandoned, sills or stringers were laid alongside the row of props and the flooring was placed on them. The props were then blasted. In mining the slice beneath, props were placed under the stringers as the room was opened out. Props were placed at 5-ft. centers; stringers were 5 × 10 × 16 ft. in length; and boards were 2 × 10 × 10 ft. in length. Figure 278 shows the arrangement.

Inclined top slicing, the Mitchell top slice, and other modifications have had their day. The scraper and mechanical loader played a part in making them obsolete. Figure 279 shows a plan for mining an ore body in radial units contiguous to an ore pass. The scraper is used to move the ore from the face to the ore pass. The numbers indicate the sequence of mining the units. A grizzly of steel rails, spaced 10 in. apart, protects the top of the ore pass. Scraper-ore movement makes possible a greater speed in mining a given unit, and may make an increase in the size of the unit possible in a given instance. The shaker conveyor may be used for short horizontal transfers.

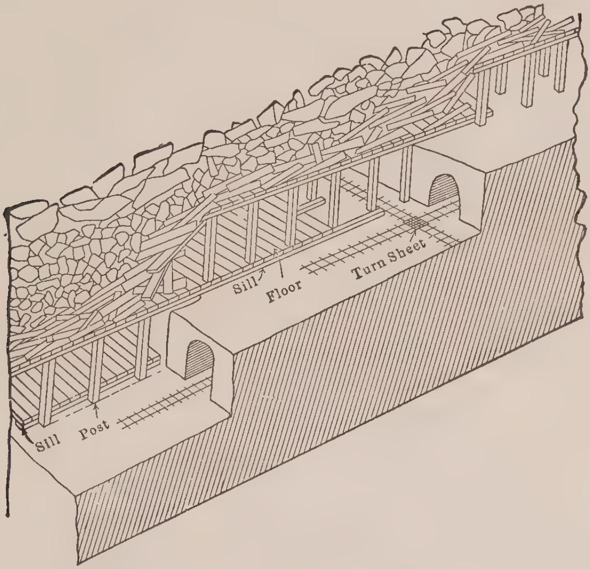


Fig. 278.—Top slicing.

7. Combined Top Slicing and Shrinkage Stoping.—The practice in the South African diamond mines best illustrates this method. The deposit is divided by sublevels into slices of much greater thickness than in the usual top-slicing practice. The interval is 40 ft. The broken "blue ground" (kimberlite) is left in the stopes, only enough to give working space being withdrawn. It thus gives lateral support and a working platform. When the stope reaches a safe length (30 to 37 ft.) the blue ground already in the stope is drawn out and the cover is allowed to cave and fill the space. The miners draw off the blue ground from the development workings. Under the protection of the blue ground in place or where it is necessary to recover ore beyond the limits of this protection, timbered crosscuts are driven through the broken blue ground. From

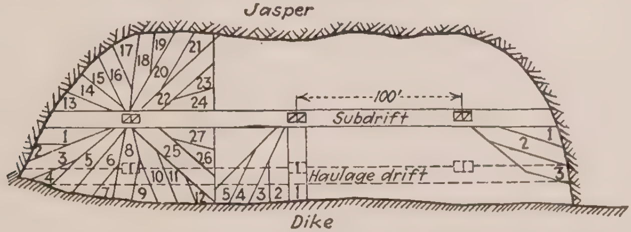


Fig. 279.—Radial top slicing with scraper handling of ore. (*Lucien Eaton.*)

the ends and sides of these the ore is drawn until waste appears, at which point the miners retreat and resume drawing at a point farther within the timbered crosscut. The method used in the Dutoitspan mine is shown in Fig. 280. J. I. Fuller's description[1] is abstracted. The stopes are carried transversely to the development workings. The first stope is started at the boundary and is carried from one side to the other. The working chamber is advanced continuously by maintaining a long inclined back, the broken blue ground furnishing a platform for the miners. Surplus ground is withdrawn through the crosscuts, which are 22.5 ft. apart. One or more of these along the length of the back are used as "pole roads" for access to the stope, while the others serve as loading points for the blue ground. The width of the stope is about 30 ft. and the length of the inclined back 120 ft. A thin pillar protects the working chamber on the side next to the cave. When the upper end of the back passes a given crosscut, the broken blue ground is loaded out until waste comes in. In order to get the last portion it is necessary to timber through the waste and draw the blue ground from the sides and ends of a timbered crosscut. Succeeding stopes can be started and follow the first, as shown in Fig. 280[2].

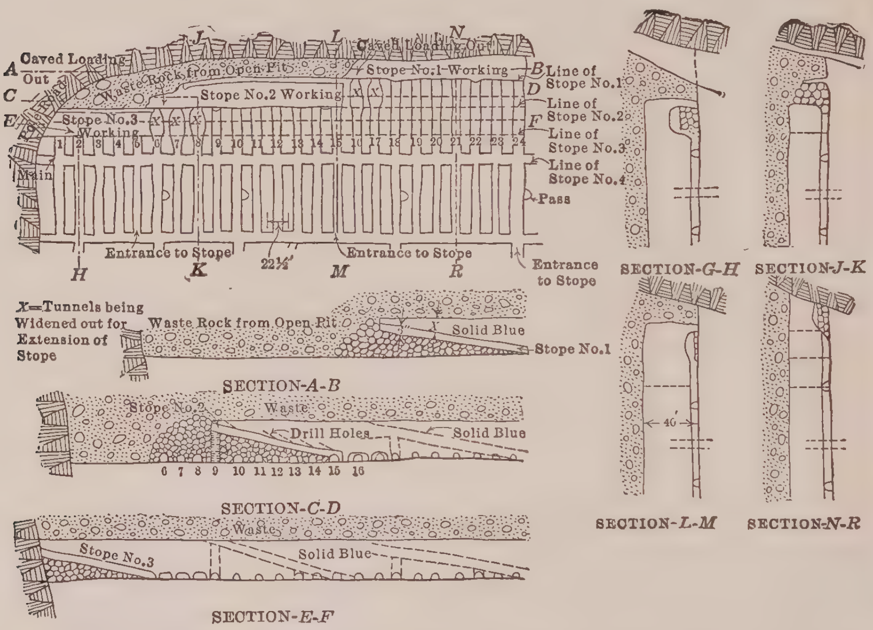


FIG. 280.—Combined top slicing and shrinkage stoping. Method used in South African diamond mines. (*Eng. Min. Jour.*)

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[1] *Eng. Min. Jour.*, vol. 94, p. 944.

[2] Mining Methods at Kimberley. *Eng. Min. Jour.*, vol. 94, pp. 887 and 943.

8. Top Slicing with Partial Ore Caving Followed by Cover Caving.—This is practicable only in the case where the cover is rock of some little tenacity. Sublevels from 12 to 15 ft. apart vertically are driven, and a system of drifts and crosscuts on each divides the slice into blocks 50 ft. square. At the edge of the ore, drifts are run connecting the crosscuts. The ore above the drift is allowed to cave and is recovered when the next drift is driven alongside the caved drift. The drifts are 7 or 8 ft. wide and are timbered and lagged on the roof. The timber is not recovered but is allowed to form a mat so as to prevent ore and overburden from mixing. The method is illustrated by Fig. 281, which represents the practice at the Newport mine.

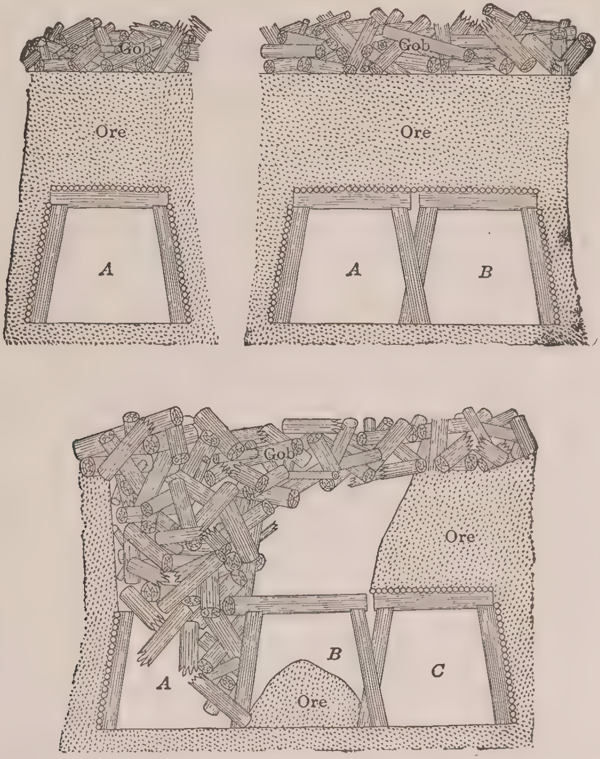
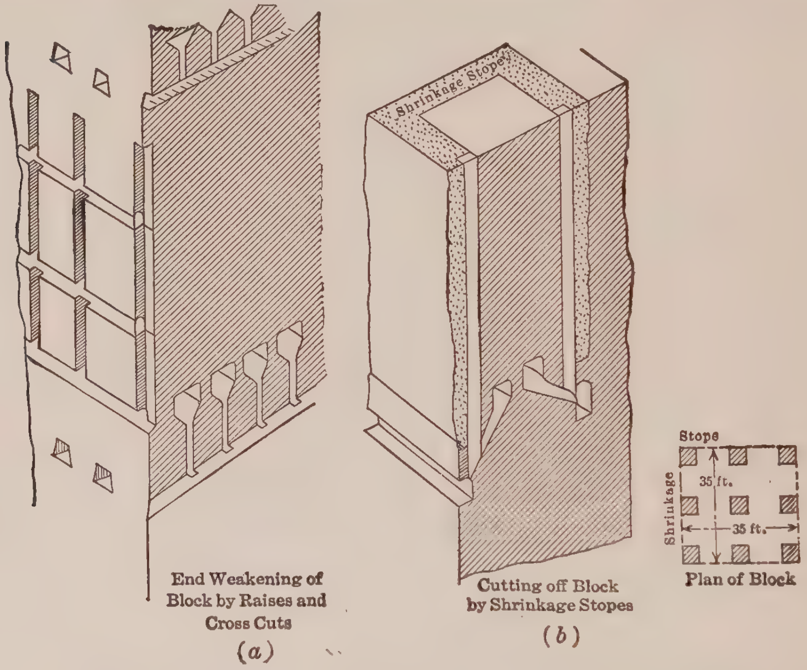


Fig. 281.—Top slicing with partial ore caving followed by cover caving, Newport.

9. Block-caving.—The method is applicable to wide, thick ore bodies but is impracticable where the ore is extremely hard and unfractured or very soft. A hard ore that has been extensively and thoroughly fractured, or a moderately hard ore that breaks readily into small pieces and when in this condition will run in a chute, is adapted to the method. Want of precise terms descriptive of the physical nature of ore body and walls as well as of extended observations upon the physical characteristics of the ore where the method has been applied prevents accurate statement concerning the limitations of this method.

The ore body is divided into blocks by levels in the vertical direction and by workings in a horizontal plane. Each block is attacked as a unit. The method is comparatively simple. Four modifications are found in practice. First, a block may be simply undercut by the intersecting workings, which leaves it supported on pillars at regular intervals. These pillars are weakened as much as can safely be done by blasting ore from them and reducing their diameter. The pillars are then drilled, and the holes are loaded with heavy charges of dynamite and simultaneously blasted. The weight of the ore block, which is supported at the ends and sides, causes it to break away and fall. In falling it is fractured and broken sufficiently to enable the ore to be handled. Second, a block may be cut off at the ends by stopes and then undercut as described above. In place of stopes, the weakening of the end by raises and crosscuts may be all that is necessary. The end weakening causes the block to break from its neighbors more readily and makes the weight of the falling mass of ore more effective in crushing the ore. Third, the block may be cut off on both sides and ends and then undercut. In place of open stopes the ends and sides may simply be weakened by drifts, raises, and crosscuts, as shown in Fig. 282*a*. Fourth, where the top slice



End Weakening of Block by Raises and Cross Cuts
(*a*)

Cutting off Block by Shrinkage Stopes
(*b*)

Plan of Block

Fig. 282.—Methods of weakening the ends of blocks in block caving.

is covered by waste that might readily mix with the ore, the mixing can be prevented by stoping the top of the ore and filling the flat stope with timber. The timber mat will prevent to a considerable extent the mixing of ore and waste from the cover. Sides and ends may or may not be cut off.

The ends or sides or both can be cut off by shrinkage stopes instead of open stopes, as shown in Fig. 282*b*. The shrinkage stope is in most cases more economical than the open-timbered stope. Whether the block is to be left attached to the end or sides, merely weakened at the ends or sides, or cut on all sides depends upon the physical nature of the ore body. An ore body already thoroughly fractured or an ore that is moderately soft and readily broken may simply require undercutting and caving, while a tough unfractured ore may require the mass to be weakened to a considerable extent on four sides.

The success of the system is dependent upon an overburden of considerable thickness that adds its weight to the weight of the ore, crushing the mass of ore as it moves downwards. A minimum width of some 125 to 150 ft. is necessary to establish an arch of sufficient dimensions for the weight to be effective in crushing the ore. Weakening the supports of the arch and the drawing off of the crushed ore cause the cave to move continuously upward. In the earliest applications of the system, timbered crosscuts were extended into the caved ore, and it was loaded out by shoveling from the sides of the crosscuts. Later, transfers with belled top were evenly spaced over the bottom area of the cave and the ore was loaded into cars by chutes, thus avoiding shoveling and timbering in caved ore. A definite system was finally evolved, consisting of an undercutting level, a chute-control horizon, a grizzly level, and beneath this a series of inclined transfers that delivered the ore to chutes on a main haulage level. Block dimensions at the Ohio mine were 100 by 100 by 60 ft. in ore thickness. At the Pewabic mine the block dimensions were 250 ft. long by 100 ft. high and the width of the ore body.

Miami caving methods developed into the one shown in Fig. 283, which established a standard practice. The unit is a block 150 by 300 ft. in area by 315 ft. in height, with a capping of approximately 260 ft. in thickness. The undercutting level is 130 ft. above the main haulage level, the base of the chute controls 11½ ft. above the grizzly level and 30 ft. below the undercutting level, and the grizzly level 100 ft. above the main haulage level. Main haulage drifts are driven transverse to the long dimension of the block at distances of 75 ft. from either end and with a 150-ft. interval between. A system of transfer raises serves the grizzly level, as shown in Fig. 283. Each raise branches to correspond with the grizzly spacing of 25 ft. On each side of a grizzly, inclined

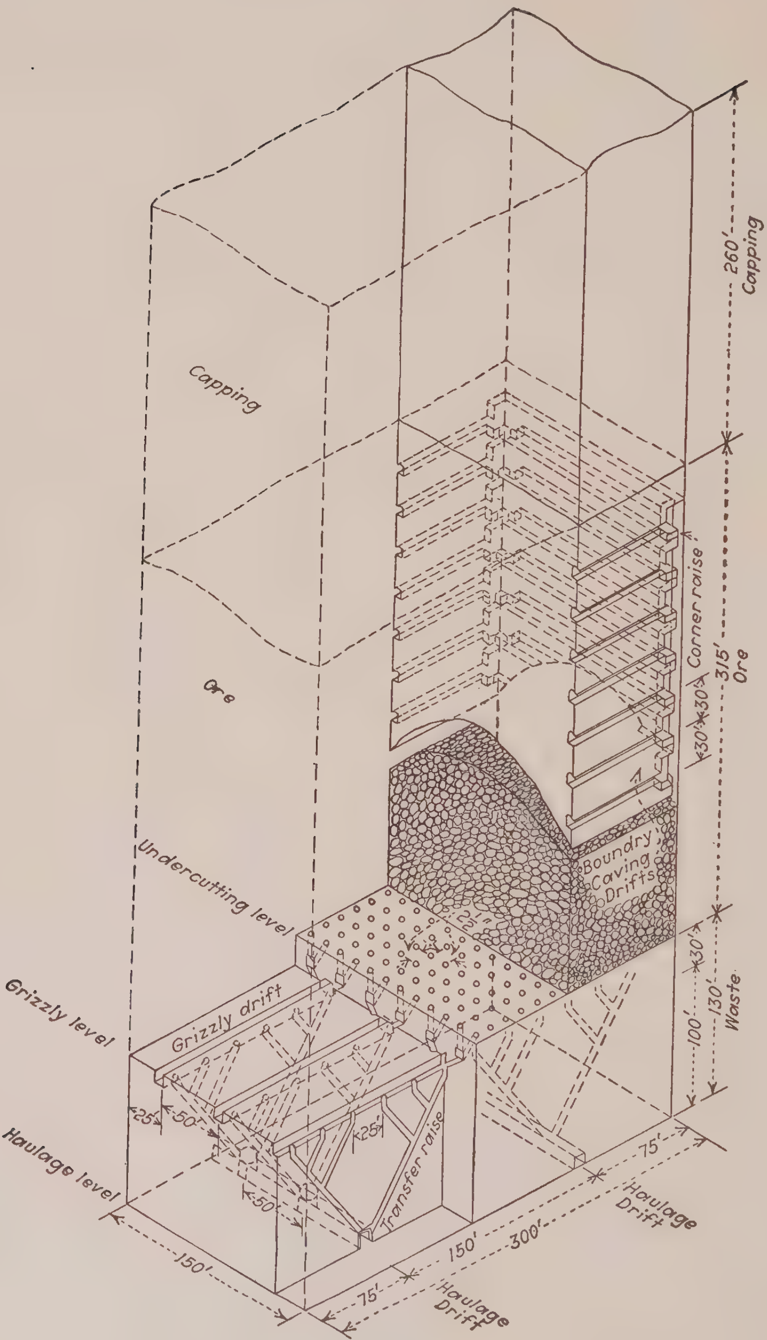


Fig. 283.—Development of a 150- by 300-ft. ore block for undercut caving. Miami mine. (*F. W. Maclennan.*)

raises terminate in a four-way chute control from which four short raises, each 4 ft. in diameter, give access to the undercutting level. The spacing of these draw points is at 12½-ft. centers in both directions. The arrangement is shown in more detail in Fig. 284. Grizzlies consist of three 45-lb. rails placed across the grizzly drift at the mouth of the transfer raise and supported on two 10- by 10-in. stringers. This arrangement gives four openings, each 10 in. wide and 4 ft. in length.

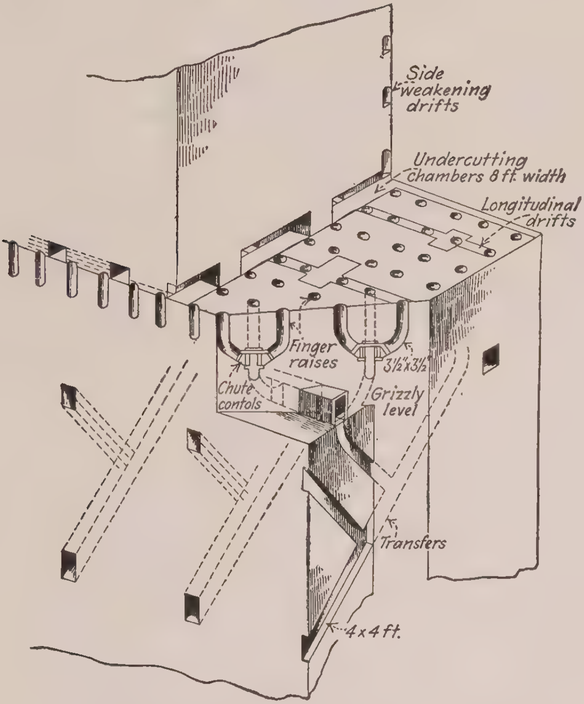


Fig. 284.—Details of finger raises and grizzly level. Miami mine.

The square sets forming the support of the chute controls are oriented so that the sides are at an angle of 45 deg. with the axis of the grizzly drift (Fig. 285).

Weakening of the initial block at the sides and ends is accomplished by driving drifts at vertical intervals of 30 to 45 ft. from a conveniently located raise, through which men and supplies reach their positions. The drifts are 4 by 7½ ft. in size and are blasted out to a height of 14 or 15 ft. above the floor. Weakening at the corners is accomplished by raises 4 to 5 ft. in diameter, which extend upward from the undercutting level to the top boundary caving level, the corner raises being enlarged by drilling and blasting a cut 6 ft. in depth out of the bottom of the caving drift for a distance of 15 ft. along both drifts from the corner.

Undercutting is started by enlarging one of the end crosscuts on the undercutting level into an opening 8 ft. wide by 22 ft. high, two successive cuts being required for this height. A. J. McDermid states that the blasting is done in four sections, each section including 37½ ft. of the crosscut and progressing across the stope in benches. On the completion of the work at the end of the stope, further undercutting is done from the longitudinal drifts by beginning at the end of the stope and opening out rooms 8 ft. in width by 25 ft. in length at right angles to the longitudinal drifts. The rooms are 6 ft. in height and are long and wide enough to connect with the four finger raises at the ends of the room. Rooms and drifts provide connections with every finger raise before undermining of the pillars begins. Horizontal holes are drilled into the ends and sides of the rooms in sufficient numbers to break up the pillars when blasted. The rooms are opened up and blasted in order across the stope, the line of retreat being in echelon. The backs of the rooms are drilled only where the ground is very hard. When undercutting reaches the end of a block, a vertical cut is made similar to the starting cut. The cutout area at the undercutting level in rooms, end stopes, and longitudinal stopes is approximately 33⅓ per cent of the area of the block. The weakening of the side walls also approximates one-third of the side area.

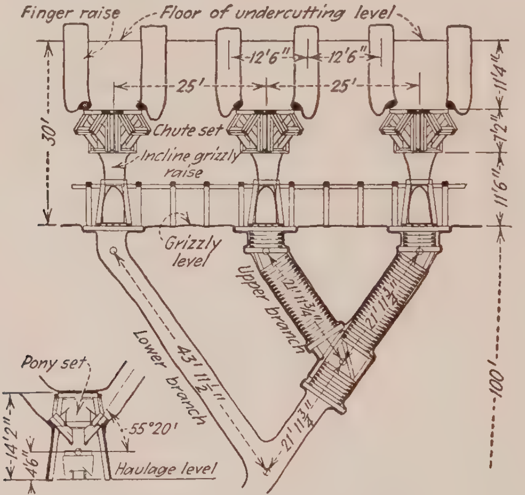


FIG. 285.—Transfer system—grizzly level—chute controls and finger raises. Miami mine. (*F. W. Maclennan.*)

Drawing of the caved ore begins on the completion of undercutting. The ore is drawn through the finger raises at the chute controls. In a stope of the dimensions given there are 72 chute controls and 288 finger raises. Eight finger raises and two controls serve a single grizzly, of which there are 36 to the stope area. Maclennan states that drawing is slow work to begin with, as the first ground caved usually arrives at the chutes in large pieces, and sometimes the stope back hangs up and drops only a little ore for several weeks, but that this is unusual. The important objectives are to draw a maximum of ore tonnage with a minimum of dilution and to regulate the drawing so as to relieve damaging weight on the extraction openings below the broken ore. At Miami, where the normal daily production was 18,000 tons, some 13 to 14 stopes were required to produce this tonnage, an average of some 1,322 tons per day being drawn from each stope, or 1 ton for each 17.06 sq. ft. of active stope area. The equivalent of solid ore is 0.733 ft. Drawing operations are under the supervision of three stope engineers (day shift), who inspect the stopes daily and issue ore-drawing orders to four draw bosses and five chute sealers on each of the three shifts. Chute sets are numbered and the four chutes of each set are lettered *A*, *B*, *C*, and *D*. The *A* chutes are drawn on one shift, the *B* chutes on the next shift, and so on in rotation, the drawing being done by two-man crews, a chute blaster, and a chute tapper. The blaster attends to all ordinary chute blasting, but a special crew is employed for difficult work. Small hand batteries are used by the blaster. The tapper crew draws from 12 to 15 finger raises per shift, blasting being necessary from 5 to 8 times per shift. The tonnage per crew is 400 tons, and normally 50 tons are drawn daily from each chute listed in the draw orders. This production is interfered with by capping, the necessity for repairs to a chute, weight on the timbers (which is relieved by drawing), ore grades, and the operations on the haulage levels. Chutes that show cap rock in advance of anticipation are sealed for several weeks. When the grade drops below a prescribed limit, drawing ceases at a given chute. By coordinating ore haulage and chute drawing, the ore drawn is measured in the cars and an exact record is obtained of the ore drawn from about 75 per cent of the finger raises. Charts are made up to indicate the condition of a stope both transversely and longitudinally. A detailed system of reports is necessary for the control of this important feature of the mining.

Maclennan gives the record of some 13 stopes completed and drawn as follows: Expected tonnage 11,038,070 tons, 1.026 per cent copper; mined 12,710,378 tons, 0.912 per cent copper; the best stope showed a tonnage extraction of 121.3 per cent, grade 97.14, and 117.81 per cent of the total copper; the poorest original stope showed a tonnage extraction of 98.28 per cent, grade 89.48, and 87.94 per cent of the total copper; the poorest pillar stope showed a tonnage extraction of 93.33 per cent, grade 81.28, and 75.86 per cent of the total copper. The 13 stopes contained 802 chute sets, 39.2 per cent of which required to be completely replaced during the operation of drawing, some of them several times;

total replacements amounted to 862, or 107.5 per cent of the original installation. In addition there were 969 minor repair jobs, though in 341 sets, or 42.5 per cent of the original installation, no repairs were required.

In the Miami example of undercut caving, 1 cu. ft. of development excavation served for about 36 cu. ft. of ore developed and ready for mining. This high ratio is one of the important advantages of undercut caving. It is evident that the height of ore tributary to a development system of the kind described would influence this development ratio. Assuming that in the Miami example the ore thickness averaged over 300 ft., an ore thickness of 100 ft. would have required somewhat less development work, but the ratio would have been about 1 cu. ft. for some 15 cu. ft. of ore developed. On the other hand, a greater thickness than 300 ft. would yield a greater number of cubic feet of developed ore. Direct mining costs at Miami averaged 40 cents per ton, of which development was 10 cents per ton, or some 25 per cent of the direct mining costs. Surface subsidence at Miami is shown in Fig. 286.

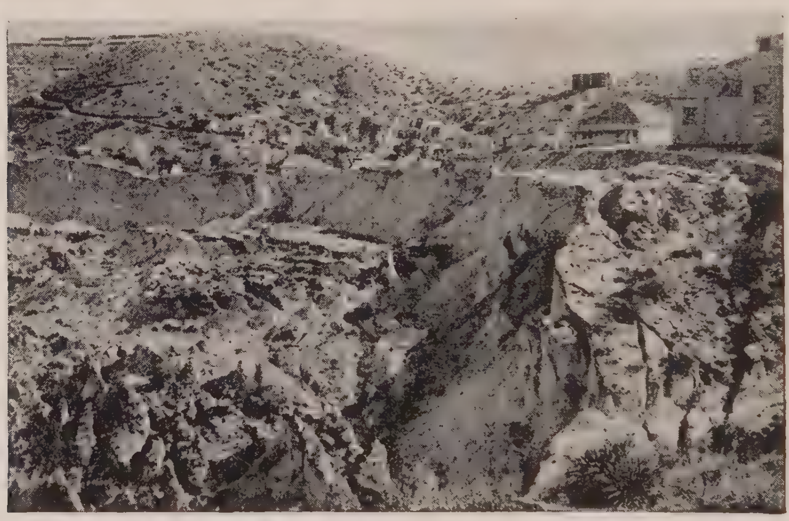


FIG. 286.—Surface effects of undercut caving at Miami.

Improvements in Block-caving.—Block-caving is practiced at Miami, Inspiration, Ray, Braden, Kimberley, Climax, Sunrise, King, Johnson, Crestmore, and other mines. It is applied to copper, iron, molybdenum, asbestos, and limestone mining. Many improvements have been made in the Miami practice just described. Chute controls are now at the grizzly level. A single set of parallel drifts is on the undercut level transverse to the grizzly drifts. A single haulageway with its main and branch transfer raises spaced on 50-ft. centers serves a block 150 ft. in width by 150 to 800 ft. in length.

At the Climax mine (Colorado) a slushing system has displaced the grizzly-branch raise transfer system, reducing development footage and eliminating chute gates and structures. R. Henderson gives a comparison between development by the slusher and grizzly-branch transfer systems. For a block area 500 × 600 ft. the slusher system required 12,875 ft. and 1,412,265 cu. ft. of rock excavation; grizzly-branch raise transfer required 22,734 ft. and 1,635,682 cu. ft. of excavation; the respective ratios are 4.71 cu. ft. and 5.44 cu. ft. per square foot of block area. Tonnage per square foot depends upon the height of the ore block, which at Climax ranges from 317 to 500 ft. above the undercut floor. Concrete is used for lining grizzly chambers and slusher drifts in preference to steel and timber. In the slusher system each slusher drift requires 741 cu. yd. of concrete, or 22,230 cu. yd. for 30 drifts; the grizzly system required 100 cu. yd. per unit and a total of 12,000 cu. yd. for an equivalent block area. Preparation of the chute and grizzly system cost 120 per cent as much as the slusher system; its operation, 118 per cent.

Haulages are at 200-ft. centers; slusher drifts are at 50-ft. centers at right angles to the haulages and 105 ft. in length on either side of the center axis of the haulage, drift floors being on a level with the top of the haulage. The scrapers discharge into steel hoppers above the haulageway into 10-ton Granby-type cars. Slusher hoists are 100 hp.; scrapers are of the hoe type, 72 in. wide, or of the half-box type, 60 in. wide; speed is 250 ft. per min. and distance 100 ft. They are placed on the steel-loading hoppers. The drift floor consists of five steel rails set in concrete and gives a smooth runway. From four to six fingers discharge into the 200-ft. slusher drift, each finger serving a stope area of 1,750 sq. ft. and each drift an area of 10,000 sq. ft. The undercut floor, which is 40 ft. above the slusher drift, is started by driving 45-deg. inclines from the fingers. Pillars are long-hole drilled and blasted in groups retreating towards the manway and from the foot to the hanging wall. Tons mined per man-shift are 31 or more; a nominal amount of both timber and steel is used; concrete requires 0.054 cu. ft. of cement per ton mined, explosives 0.5 lb. per ton, and power used 3.04 kw. per ton. Normal tonnage per day is 15,000.

Practice at Emma Nevada mine (Nevada) shows many variations. Conventional methods were first applied in 1927. When mining was resumed in 1937 after a shutdown period, slushers were introduced and other caving details were improved. It is not possible to cover all the variations, but the cross-slushing layout used for high-lying ore bodies is of interest here. Haulage drifts are at 180-ft. centers; slushing drifts are at 30-ft. centers; finger raises on opposite sides are spaced at 15-ft. centers. The slusher drift, which takes the place of the grizzly drift, is some 240 to 270 ft. in length. Short vertical passes at the end of the slusher drift connect with a cross-slushing drift, which in turn connects by inclined transfers to chutes on either side of the pony set on the haulage level. The primary slusher serves 16 fingers; the cross or secondary slusher serves three primary slusher drifts and one inclined transfer. A stope area 90 × 240 ft. is served by three primary slushing drifts and one secondary or cross-slushing drift. In another arrangement cross-slushing drifts are omitted and branch transfer raises receive the primary slusher discharge. The objective in both arrangements is to reduce development footage as far as possible by resorting to horizontal scraper transfer or the use of shaking conveyors for the same purpose.

Miscellaneous Stoping Practice.—In 1925 W. C. Browning and F. W. Snow described mining methods in use at the Magma mine (Arizona). Primary development was in the footwall. Short crosscuts were driven to and across the ore body at 150-ft. intervals, from which footwall drifts were extended. The ore body stood at a high angle and varied in width from 10 to 45 ft. The standard level interval was 100 ft., though in some instances an interval of 200 ft. was taken. Stope lengths were 60 ft. for a 10-ft. width and 30 ft. for a 15-ft. width. Sill floors of the stope were square-set. The sets at the hanging-wall side were extended upward at the center of the stope and raise cut in the hanging wall to the level above to serve as a ventilation and waste pass. The waste rock was used for stope filling. The stope was extended upward as a timbered rill (inclined cut and fill with square sets) with open square sets at each end of the stope, one for extraction and the other for a manway. After each filling, floors were laid upon the slope, 35 to 40 deg., and a 7-ft. slice of ore was broken down; two additional slices were taken before the next waste-filling cycle. Succeeding stopes on either end of the first were started, their lengths being 15 ft. for wide sections and 30 ft. for narrow parts of the ore body. Figure 287 shows the method.

A combination rill and underhand pillar (Mitchell slice) was also described. The units were 15 ft. along the strike and from 15 to 45 ft. across the ore body or the width of the ore. Two rill stopes, each 15 ft. in width and separated by a pillar, were carried to the level above and were filled with waste. The intervening pillar, 15 ft. in width, was cut across at the top, stringers and knee braces supporting the level above. The pillar was mined underhand, the broken ore being drawn off at either side through the open sets; the side sets were supported by stringers placed across the stope. On completion, the open stope was filled with waste, the stringers being retrieved as the fill rose. Cable ties were placed in the rill stopes that bound vertical posts supporting lagging boards alongside the pillars. In this way the fill was prevented from running into the pillar stopes. The broken ore from the stopes was drawn off by a system of inclined passes in the footwall, 52-deg. slope, as shown in Fig. 288, section *c-c*. In silling out the rill stopes, scrapers were used to bring the ore to the passes until the filling slope had been established.

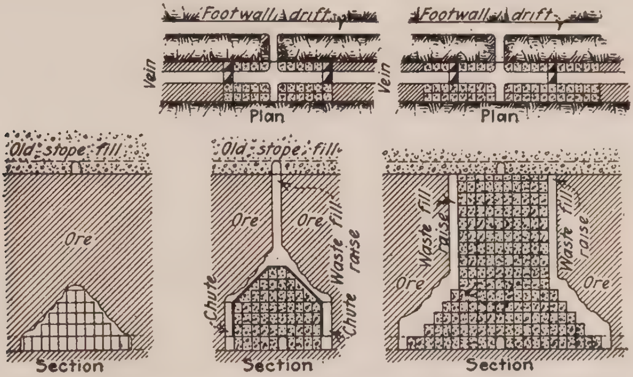


FIG. 287.—Details of timbered rill stopes. (*Eng. Min. Jour.*)

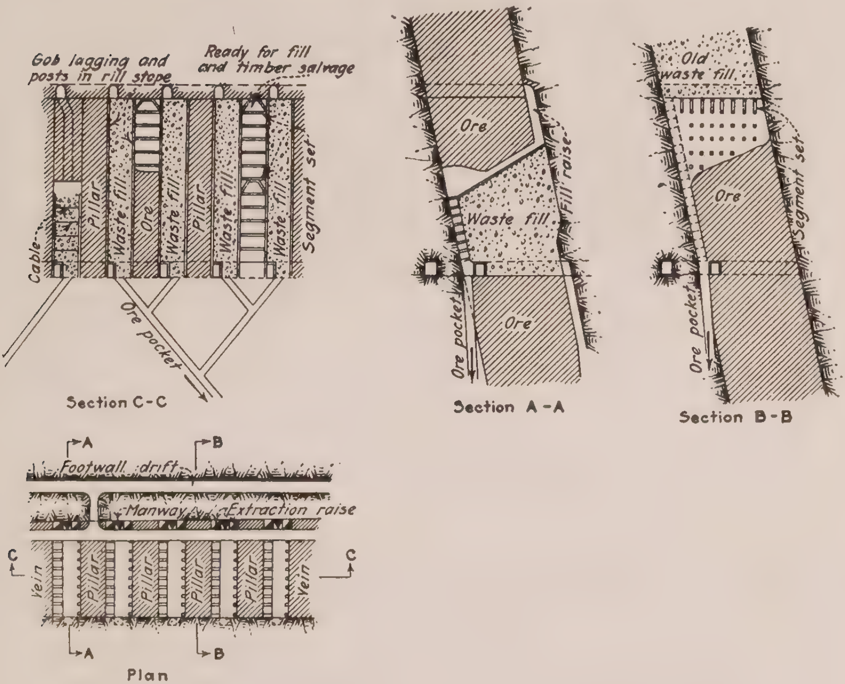


FIG. 288.—Combination rill stoping and Mitchell slice. (*Eng. Min. Jour.*)

Figure 289 shows the details of a Mitchell stope, described by J. B. Pullen, as applied at the United Verde mine. The parallel slots are timbered with square sets, 6 × 6 × 8 ft. in height, 10- × 10-in. timbers, and are driven on 18- to 24-ft. centers to the height of the stoping section.

The length of the stope is limited to 60 ft. and the height to 54 ft., the average stoping section being 48 ft. in length and 41½ ft. in height. Mitchell stopes were used at the United Verde in mining horizontal and vertical pillars left after mining by horizontal cut-and-fill stoping. The advantage over all square sets is that 50 to 60 per cent of the ore can be mined with but very little timber, as many of the stringers may be retrieved.

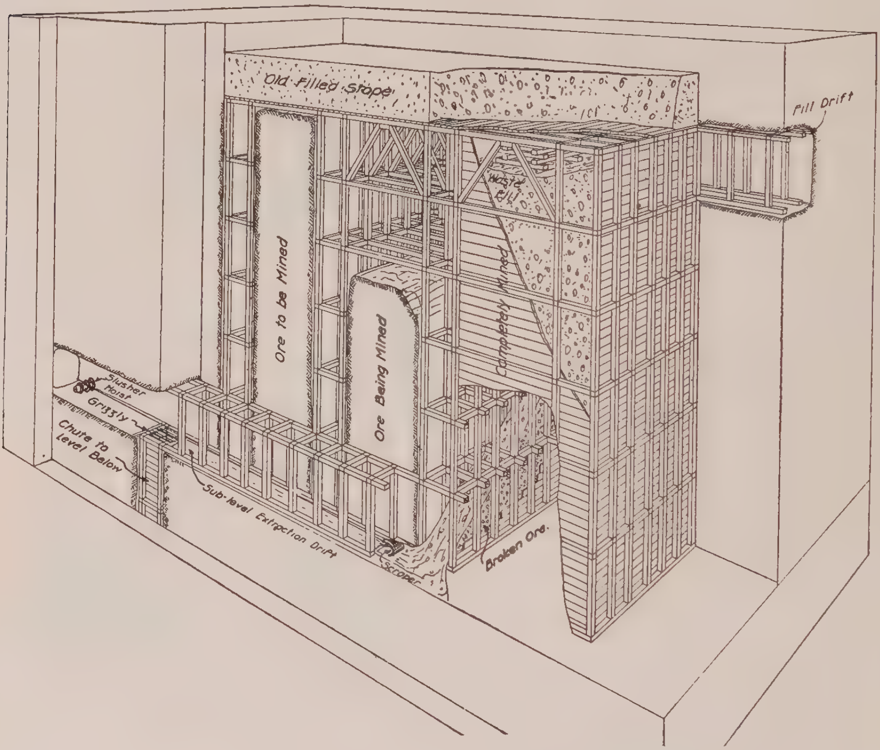


Fig. 289.—Mitchell stope. (*A.I.M.E.*)

Figure 290*a* shows a method similar to the Mitchell stope. It was described by C. A. Kumke, who called it the "slot system." It was used at the Golden King mine (California), where stope backs would not arch and sustain themselves in shrinkage stoping. The slots were narrow square-set stopes extending from footwall to hanging wall or to the stope-wall limits as established by sampling, and from one level to the level above with a vertical ore pillar some 30 ft. in width between. The square sets were tied at intervals by means of used ⅜-in. cables and in addition were fastened to anchor bolts in the pillar walls. Unlike those in the Mitchell stope, the pillars were mined by shrinkage stoping.

Sublevel Stoping and Diamond Drilling.—Sublevel stoping at Noranda mine (Canada), where stope heights were 175 ft., widths 60, 75, 90, and 114 ft., and lengths 300 to 350 ft., called attention to the advantage of long-hole drilling of such stopes. The stopes are in strong ore, separated by 40-ft. pillars. Sectional drill steel was first used and then light diamond drills and diamond-drill holes. Sublevel intervals were increased to 35 ft. from the usual interval of 25 ft. This reduced the proportion of development volume relative to ore volume. In some stopes the sublevel interval was increased to 60 ft. and a pair of sublevels, one on each side of the stope. The 114-ft.-wide stopes were provided with an additional line of sublevels in the center axis of the stope. This provided for a maximum depth of 20 ft. for blastholes. By increasing drill-hole depth to 45 ft. only two lines of sublevels were required for a stope width of 90 ft. (see Fig. 290*b*).

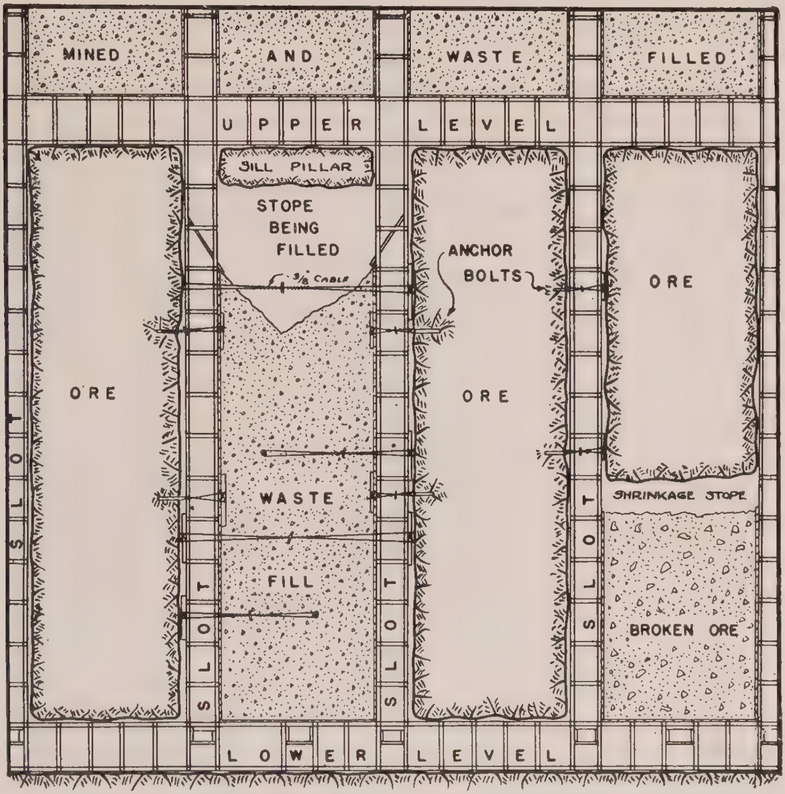
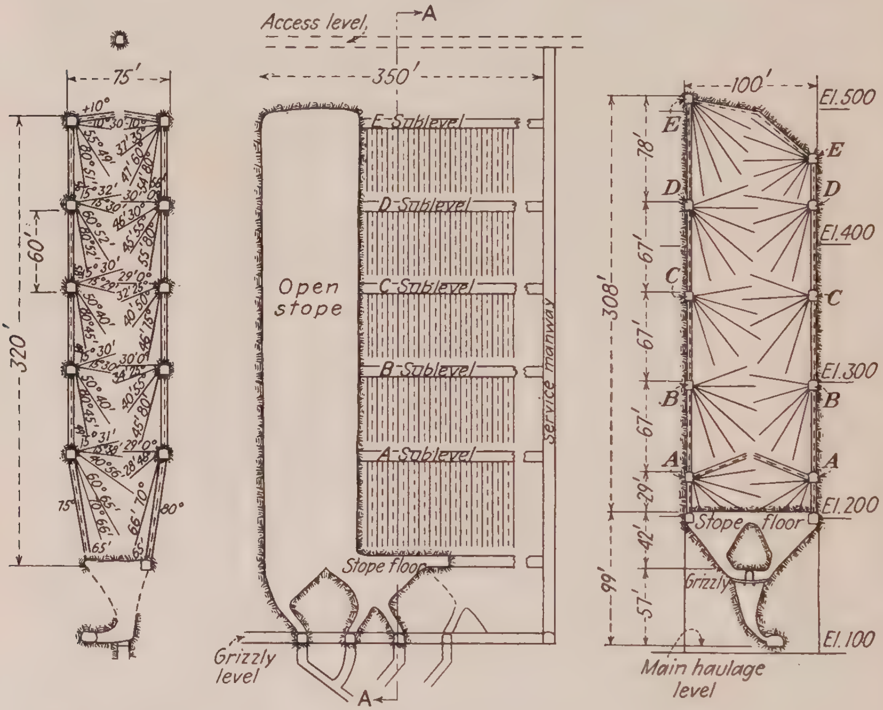


FIG. 290*a*.—Pillar mining by shrinkage stoping. (*A.I.M.E.*)

Drilling depth is an important factor in determining the position, spacing, and number of vertical sublevel rows for a given stope. For a maximum hole depth of about 65 ft. the sublevel spacing was 60 ft. In other mines the stoping height was divided into two 100-ft. lifts, each served by a sublevel from which diamond-drill holes were fanned out. Down holes are preferred; holes of greater upward slope than 15 deg. are avoided, as uppers are more difficult to load. The width of the ore body, its regularity, dip, and physical characteristics determine the layout of the stope and its development.



CROSS SECTION A-A SIDE ELEVATION

Fig. 290*b*.—Sublevel stoping with diamond-drill holes for blasting charges. Left and center—Noranda rhyolite flux stope. Right—Waite Amulet sublevel stope. (*O. V. Lindquist.*)

Porritt and Carmichael state that at Noranda a typical blast of some 12,000 tons entailed blasting 80 holes aggregating 9,503 ft. in length. The total labor for cleaning out holes, loading, testing circuits, and connecting up was 115 man-hours.

The light diamond drill, improved drill bits, electric blasting, firm walls, and large ore bodies made Noranda sublevel practice possible. Ground movement was prevented by filling stopes with mill tailings and granulated slag after they were finished and emptied. The presence of pyrrhotite in the mill tailings resulted in the cementing together of the fill material.

Open-stope and Diamond Drilling.—The successful mining of pillars by diamond drilling and long-hole blasting no doubt suggested open-stope mining by the same method. There are many examples of this. In some, horizontal ring drilling is effected from the corners of the stope area; in others horizontal parallel holes are drilled from slots at the ends of the stope, or parallel holes, inclined or vertical, are drilled from benches at the top of the block. Most of these applications are retreating from the bottom upward, the broken ore falling into hoppered bottoms connecting with loading chutes or in many cases with bulldozing chambers, where secondary blasting can be accomplished.

Shrinkage Stoping and Diamond Drilling.—Mounted drills and horizontal holes are preferred to stopers in shrinkage stoping. Such stopes should not be less than 8 ft. in width, as arching may develop in narrower stopes. Both walls and ore back must be strong or self-sustaining. In wide ore bodies the light diamond drill and long blastholes have found some application.

Parallel horizontal holes are drilled from positions slashed out at the ends of the stopes. Holes are spaced about 6 ft. apart and burdens are 4 ft. or more. It is unnecessary for the miners to enter the stope. At the Sladen Malartic mine (Canada) slots or cuts were ring blasted at the ends of the stope, giving a breast 13 to 14 ft. in height; diamond-drill holes were next drilled from these breasts 50 ft. in depth, spaced 4 to 6 ft. horizontally and with a 4-ft. burden on each horizontal row; the slice blasted down was 50 to 60 ft. in length, 10 to 30 ft. in width, and 12 to 14 ft. in thickness; drill footage gave 1.25 to 2.5 tons per ft. and explosives were 0.35 to 0.5 lb. per ton.

Shrinkage Stoping and Mechanical Loading.—Shrinkage stoping requires drawoff raises and chutes uniformly spaced at intervals of from 25 to 30 ft., and wide stopes may require two lines of such chutes to serve the area. In comparatively recent practice drawoff raises are started from the floor of the level and the chute construction is omitted. The broken ore is loaded out with mechanical loaders. Crosscuts are driven from a main drift at intervals of 30 to 50 ft., and from these the drawoff raises are extended up to the stoping level in wide orebodies. In all mechanical loading, drawoff points must be offset from the haulage drift a sufficient distance to give room for the loaders.

Spiral Stoping.—Spiral stoping was developed at the Granby Company's Phoenix and Anyox mines. It is applied at the Beattie mine (Quebec) to ore widths 60 to 200 ft. and stopes 100 to 110 ft. along the strike, with intervening 30-ft. pillars normal to the strike. Levels are at 150-ft. intervals. Each stope is served by two chute raises, which have 50-ft. centers and are 25 ft. from the nearest pillar. A manway between the chutes connects with a short drift joining two bulldozing chambers above the chute pockets. Incline raises start at a 40-deg. slope from the edge of the grizzly opening. Manway raises are constructed in the pillars, zigzagging up the center line of the pillar. The spiral stope on a 35-deg. incline turns about an axis with a 35-deg. slope on the inside of the turn for an ore runway and a flatter slope on the outside for a pathway. Connections are made with the pillar raise. Successive slashes are taken from both back and sides. Machine shift tonnage increases from 50 to 150 tons in successive slashing stages.

10. Longwall Method.—The longwall method is one of the important methods of mining coal. It is not restricted to coal but can be extended to the mining of sedimentaries such as clay, gypsum, and salt. Unlike other methods of coal mining, the longwall method wins all the coal on the first working. Coal seams from 2 to 8 ft. thick and from 0 to 30 deg. dip are worked. An essential is that the roof must not be too hard. The characteristic of the method is a continuous working face, which may be straight or stepped, followed by a uniform settling of the roof behind. The working face is divided into sections 40 ft. in length, each section being served by a gateway or branch passage equipped with track for tramming. The sections may make a continuous face, either curved or linear. They may be stepped where a heavy roof must be handled. The coal is removed by undercutting and blasting, by undercutting and allowing the roof pressure to bring down the coal, or by roof pressure assisted by more or less picking. The weight of the roof is carried in part by the working face and assists in getting the coal. A sufficient area must be opened up in order to get the roof working properly and to start the initial break at the shaft pillar. In some cases the roof weight becomes excessive and must be controlled by timber cribs and gob packs. The face is protected by three or four lines of props, which are placed from 3 to 4 ft., center to center. As the face advances, the last line of props is removed and a new line is placed. Waste is thrown back into the gob. The gateways are protected by waste packs on each side. Where a narrow seam is being extracted, headroom in the gates and at the faces is obtained by "brushing" the roof or floor (excavating 2 or 3 ft.) The waste so obtained is used for pack walls and stowing. Where the roof pressure is too much for props, timber cogs are used instead. The longwall face may be advanced from the shaft pillar (longwall advancing), in which case gateways, mother gates, and main haulageways must be extended through and maintained in the gob; or the development workings may be pushed to the boundaries of the property and the longwall face brought to the shaft (longwall retreating). Longwall advancing is more often used, as it reduces the time required for development and thus brings a mine to the producing stage sooner. The panel system may be used. This requires that the coal seam be divided into panels 600 by 1,200 ft. or other dimensions. Each panel is protected by barrier pillars, usually 50 ft. wide on all four sides. Within the barrier pillars the coal is removed by a longwall face, which may be extended diagonally or parallel with the ends of the panel. The longwall face is always brought back to the main development workings. In large mines the panel system is probably the best method.

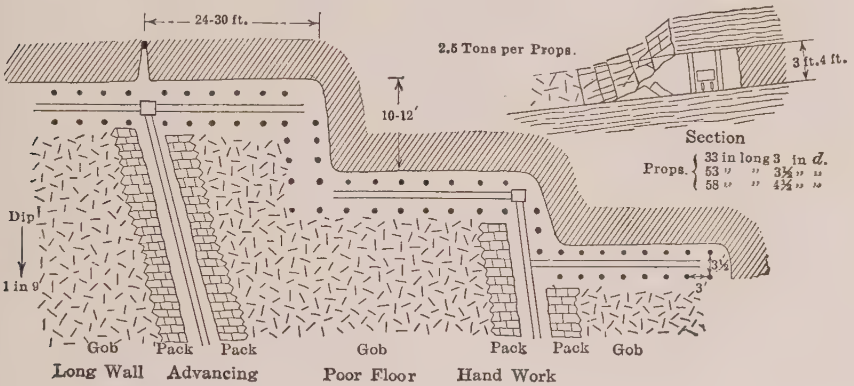


Fig. 291.—Longwall advancing.

The development plans show how easily ventilation is accomplished. The coal is shoveled into cars, which may be run on a track parallel with the face, or each 40-ft. section may be served by a stub track intersecting its center. Longer sections can be served by putting in turnplates and sections of track parallel with the longwall face.

Figure 291 shows plan of a stepped longwall face with gates at 60-ft. centers.

11. Room-and-pillar Method.—The following names are in use: "Room and pillar," "pillar and breast," "post and stall," and "bord and pillar." The first three have practically the same signification. Stall as distinguished from room is characterized by a narrow opening, whereas the ordinary room may open out from an entry with the full width or less. The fourth, bord and pillar, has features that distinguish it from the room-and-pillar method, and consequently it is treated separately in the section that follows. While the room-and-pillar method is used principally for mining coal, it can be applied to the mining of any mineral that occurs as a bedded deposit. The following description applies particularly to coal deposits. Coal seams ranging from 4 to 12 ft. thick and from 0 to 30 deg. dip and at a depth not greater than 500 ft. can be worked by room and pillar. A narrow room is excavated, starting from a side or butt entry. This may be driven at right angles to the entry or inclined at any angle from a right angle to 45 deg. A neighboring room is taken off, separated by a rib or pillar. As the rooms are extended, short drifts or break-throughs are driven through the ribs for ventilation. Other rooms are started in sequence. The rooms are 250 to 300 ft. long and extend up to a pillar that separates them from the next side entry. A track is laid in the center of each room. The coal is loaded into cars by shoveling. As soon as a number of rooms on one entry have reached completion, work is started on the ribs between the rooms, beginning on the rib between rooms 1 and 2. This rib is worked back toward the entry, and as soon as it is well advanced the next rib is started, and so on. The ribs are worked back in echelon. The method of drawing the rib is to start a drift through the rib, leaving a protecting pillar on the end. The drift is timbered by props and headboards. The protecting pillar is then attacked and removed in slices parallel with the drift. Temporary props are put in. When as much of the protecting pillar as can be removed with safety has been removed, the props are drawn, the roof is caved over the original room, and the portion is mined out in the rib. Rib drawing can be started at any point in the mine, provided the work is started at the ends of the completed rooms and is extended (retreating) toward the side entry from which the rooms have been driven. Two methods are practiced. In one the mine is roomed out to the boundaries and a general retreat upon the ribs follows; in the other, rib drawing is begun almost as soon as two or three rooms have been completed, following closely and systematically upon the rooms as they are completed. The latter method is by far safer and better, and yields more coal. In determining the width of room and pillar two tendencies of practice are to be noted. In one the room is made as wide and the rib as narrow as possible, primarily to secure as much coal as possible on the first working, even at the sacrifice of the coal left in the ribs. This policy makes the drawing of the ribs dangerous and uncertain, for the ribs may be too narrow to support the roof, and creep and crush may cause the mine to be lost. This practice is, on the whole, to be condemned. The other tendency is to leave the ribs of ample width and to secure only a moderate amount of coal on the first working. The remaining coal can be won on the second working or rib drawing.

Two methods are in use for the general layout of the system. In the first the mine is worked as a complete unit, pillars being used only to protect the main and side entries. In the second the mine is divided into large panels, and each panel is protected by barrier pillars and worked as a single unit. The latter method is the better practice; the former is of use only in small mines. The primary reason for panel working is the prevention of squeezes and the localization of accidents, principally explosions. Room-and-pillar dimensions are given in the following table:

TABLE 118.—ROOM-AND-PILLAR DIMENSIONS

Length of room	Minimum, 100 to 150 ft.
	Common, 250 ft.
	Maximum, 300 ft. (Where rooms connect, lengths run up to 800 ft. or more)
Width of room	Minimum, 12 ft.
	Common, 20–25 ft.
	Maximum, 30–40 ft.
Width of rib	Minimum, 10–15 ft.
	Common, 15–25 ft.
	Maximum, 32–72 ft.

As an example, the room and rib widths given by H. N. Eavenson and representing the practice in the mines of the U.S. Coal & Coke Co. are quoted below.

TABLE 119

Depth of cover, ft.	Width of rooms, ft.	Width of ribs, ft.
Up to 300	18–24	42–36
300–500	18–24	57–51
Over 500	18–24	72–66
Up to 300	36–40	44–40
300–500	36–40	54–50
Over 500	36–40	64–60

Rooms are driven face-on, end-on, or at an angle to the main cleat of the coal. The significance of the terms is illustrated by Fig. 292. The direction of the room is more or less governed by the dip of the seam. With a steep dip the room is driven at an angle in order to reduce the grade. Where the dip exceeds 6 or 8 deg. the rooms are usually driven up the dip and the cars are drawn up to the working face by gravity trams. Two tracks are used, the loaded car hauling the empty up. In some mines the room direction with reference to the cleat has an important influence upon the production of the maximum amount of lump coal. Rooms driven face-on produce more lump coal than others. The support of

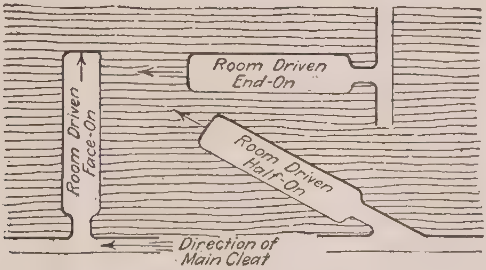


FIG. 292.—Directions for driving rooms in the room-and-pillar method of mining.

the roof is still another factor. Roof breaks parallel with the cleat can be controlled more readily when the room is driven normal to the direction of the break than otherwise.

Illustrations of room-and-pillar mining are given in the chapter on Development. Machine cutting is discussed in the chapter on Rock Breaking. Respecting the general method of procedure in coal mining, H. N. Eavenson makes the following significant statement.

> The first principle of any economically planned and managed coal mine should be to drive rooms only as they are actually needed and to begin removing pillars as soon as the rooms have reached their limits, and it is unfortunate that the limitations of output, labor supply, dirty coal, etc., too frequently prevent this being done, even where so planned.

12. Bord and Pillar.—The distinguishing characteristic of this method is the winning of a relatively small proportion of the coal in the first working. This may range from 15 to 35 per cent. In the room-and-pillar method the first working yields from 50 to 80 per cent. Bords or rooms are driven face-on and are from 12 to 18 ft. wide. They are spaced at varying distances, depending upon the depth of the workings. Shallow workings have wider rooms and closer spacing, while in deep seams the reverse practice is followed. In the Pittsburgh district room centers are 42 ft. under the former conditions and 80 ft. under the latter. Drives (walls or headings in English coal-mining practice) at right angles to the bords are put through at regular intervals and divide the coal into square or rectangular pillars. The drives are narrow, about one-half the width of the bords. Redmayne gives the following pillar dimensions for different depths:

Depth 300– 600 ft.	pillars 90 × 120 ft.
Depth 600– 900 ft.	pillars 120 × 150 ft.
Depth 900–1,200 ft.	pillars 120 × 180 or 150 × 150 ft.

The second working or the removal of the pillars may start at almost any point, but preferably at a boundary of either the panel or the mine. In the Pittsburgh district the working of a rib or pillar requires the driving of an 8-ft. chamber, 8 ft. from the end of the bord and across the rib. The narrow rib left is then worked back toward the bord in two or four sections. As each section is removed the props are drawn and the roof is caved before the next section is attacked. When the last section is reached the end of the bord is dropped together with the section. The miners start another 8-ft. chamber 8 ft. from the new end of the bord, and the working of the narrow rib left follows. Props are used for the temporary support of the roof, and track is laid for the transport of the coal. Figure 293 shows the sequence of steps involved in working a rib.

The general layout of the method follows that used in the room-and-pillar method. The bord-and-pillar method has the advantage of giving less trouble from excessive roof and floor pressures and yielding a higher percentage of the coal. It is applicable to deeply buried seams and can be applied to shallow seams where the room-and-pillar method is unsatisfactory. It is evident that the line of demarcation between the two methods is not sharp.

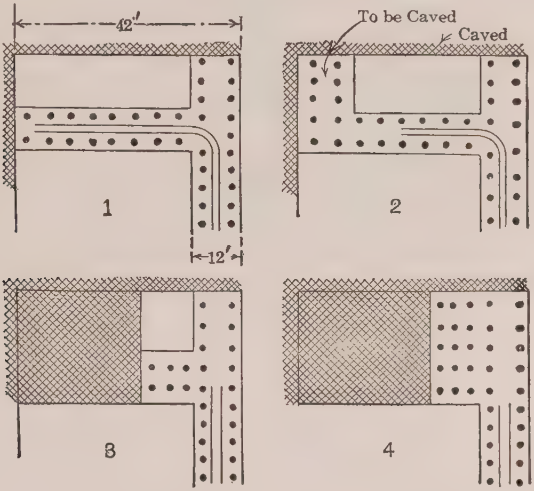
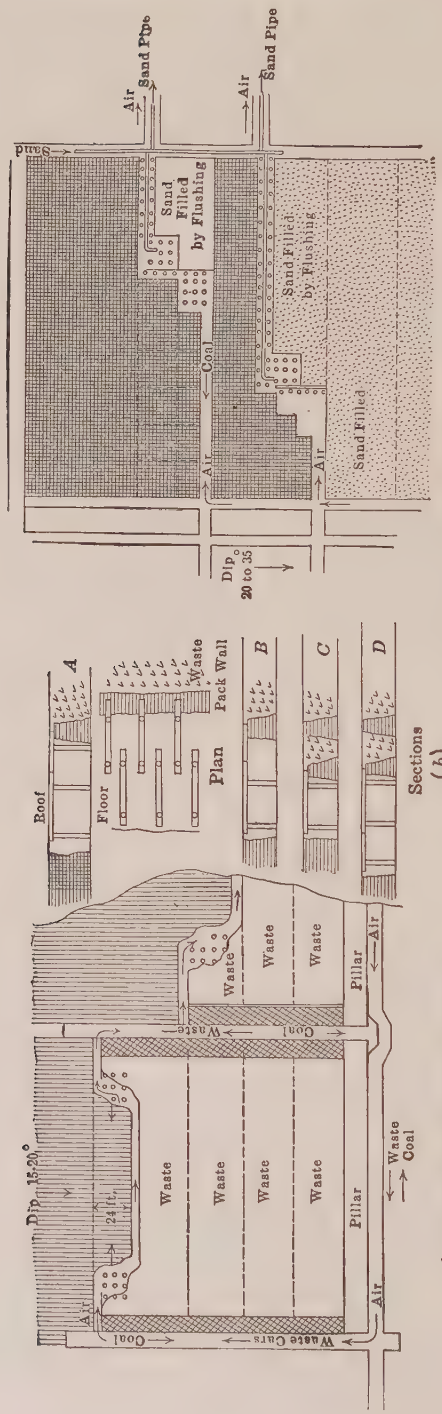


FIG. 293.—Method of working back a rib. (*Trans. A.I.M.E.*)

13. Special Methods. *Mining Coal Seams of Moderate Dip.*—The coal seam of moderate thickness can be mined by longwall or by room-and-pillar methods. Usually the longwall faces are driven up the dip and provision is made for raising empty cars and lowering loaded ones with a hoist, go-devil, or windlass. Figures 294*a*, *b*, and *c* show two methods used in the Westphalian coal district, Germany. Filling is used in both, and the placing of filling follows closely upon the excavation of the coal. As is shown in Fig. 294*a*, the filling is hoisted in cars up an auxiliary slope and the cars are run along the working drift to the point where the filling is to be placed. The track extends to the coal face, where the empty cars are filled and returned. The drift is extended until both faces driven from opposite sides meet. On the completion of the filling the track is removed and a new drift 24 ft. up the dip is started. The details of filling and roof support are shown in Fig. 294*b*, *A*, *B*, *C*, and *D*. Figure 294*c* shows the working of a seam from 20 to 35 deg. in dip and the use of sand filling by flushing.

For mining coal seams of steep dip, methods similar to overhand stoping may be employed. The room-and-pillar system is also used; chutes are constructed in the center of the room and the coal is drawn off



(a) Plan Upon Plane of Seam

Sections (b)

(c) Sand Flushing on Inclined Seam

FIG. 294.—Methods of mining inclined coal seams.

by gravity from a level below in much the same way that ore is handled. Ventilation is provided by a auxiliary airway driven as a drift in the coal above the main tramming level. Brattices bring the air to the working places in the rooms. In working up the dip, rooms are driven only moderate distances as compared with ordinary room-and-pillar work. Robbing of pillars is accomplished by retreating from the end of the levels. The coal is worked from the top of the pillar downward.

Mechanization of Coal Mining.—The first stages in the mechanization of coal mines involved the use of the power drill and coal cutters of various types. A relatively long period followed, during which the use of these machines spread and drilling and undercutting passed generally from the realm of hand labor. From about 1920 began another broad development of mechanization, principally in the use of mechanical loaders and a more extensive use of different types of conveyors. The early part of this period was marked by the mechanical improvement of the machines themselves. Design and materials used in construction have finally reached a point where dependable machines with standardized parts are available. Along with a more widespread experience with the machines came appreciation of their possibilities. The third stage in mechanization is now in progress: The improvement in mine layout and mining method and the coordinating of all operations having to do with the mining, loading, and transportation of coal. Car sizes have been increased; car design has been improved; locomotives have been made more dependable; auxiliary hoists of various types have been better designed; electrical and lighting equipment are available; and working crews have been trained in the proper use of the equipment. Ventilating systems have also been modernized.

Machine appliances have resulted in a higher rate of working and permit a greater concentration of working places, so that the same or greater tonnage is now secured from fewer working places. Supervision is thus made more thorough and roof control has been simplified. Undercutting, drilling, shooting, loading, gathering, timbering, track work, and haulage have increased in efficiency.

Coal seams of 6 ft. or greater thickness permit the use of mechanical loaders and pit-car loaders; seams of from 4 to 6 ft. in thickness permit of low-clearance mechanical loaders and pit-car loaders, but scrapers and conveyors are more generally used; while in seams of less than 4-ft. thickness conveyors and scrapers are more adaptable. Scrapers require a comparatively smooth, hard footwall for satisfactory operation. The thickness of the coal seam, its dip, the roof conditions, and the distribution of impurities in the seam are factors in the selection of the loading equipment.

Mechanical loaders used in coal mining are the Joy, Coloder, Goodman power shovel and for rock work the Myers-Whaley and Nordberg-Butler. The pit-car loader is a short length of chain conveyor mounted upon a two- or four-wheel truck, the receiving end being at track level or lower and the discharge end at car height. It is motor-driven. The scraper loader is a sloping steel pan mounted upon a four-wheel truck and equipped with motor and hoist for operating the scraper. The discharge end is at car height and the receiving end at floor level. Pit-car loaders require hand shoveling, but mechanical loaders and scraper loaders avoid hand labor.

Conveyors are of two types, the sectional chain conveyor and the shaker, or roller, jigging conveyor. The first consists of a bottom pan for the return of the chain and an upper pan for the conveying of the coal. The sections are 6 ft. in length and the chains are equipped with steel flights. The shaking conveyor is operated by a head motion and motor. The length of stroke is adjustable to conditions and ranges up to a maximum of about 11 in.; the number of strokes ranges from 60 to 82 per min.; motor size ranges from 5 to 25 hp., a 5-hp. motor being used for a conveyor 200 ft. in length, No. 1 trough, and a 7½ hp. for 300 ft., No. 1 trough; conveyor lengths up to 500 ft. have been used. Chain conveyors are used up to a length of about 300 ft. Both types of conveyors are usually loaded by hand shoveling. In the shaker conveyor an attachment, known as the "duckbill," reduces hand shoveling to a minimum. The duckbill is given the same reciprocating motion as the conveyor trough and, by means of a telescopic section, it can be advanced into the broken coal and will load it out. The Jeffrey Co. has a "conveyor loader," which is used on long-face work. The coal is shot down upon the conveyor in part, about half of the coal being retrieved without shoveling.

At Hanna, where Joy machines are used in loading out the lower 8 ft. of a 30-ft. coal seam, the loader units averaged 65 tons per machine day, and in later work with improved units an average of 200 tons per machine day or better could be obtained. The maximum with the older-type machine was 150 tons per day; with the new machines a maximum of 260 tons could be obtained. The upper 20 ft. of the seam is loaded with Thew electric shovels, which average 169 tons per day with a four-man loading crew.

G. B. Pryde states that with a crew of 16 men a scraper unit averages 326 tons per machine day and reaches a maximum of 500 tons. The scraper unit is used for long-face working and discharges into a conveyor. The usual procedure is for a scraper unit to remove the broken coal to a conveyor; the first conveyor discharges into a second at right angles to the first, the second conveyor delivering to the cars on the entry. Pryde states that the scraper can be used only in places where the roof is particularly strong, so that a span of at least 10 ft. can be left between the last row of props and the coal face. G. B. Southward gives an example of scraper loading in a seam 7½ ft. thick in which a 300-ft. face was worked, the depth of cut being 8 ft. With 14 men on the crew the output was 270 tons per shift. The units worked were 90 ft. in width, the roof being supported upon timbers while advancing the face from 75 to 90 ft. Total man-hours per ton was 0.386. The scraper loaded into cars. In pillar-recovery work where rooms were 22 by 300 ft. and pillars 50 ft. wide, a 10-man crew, making a cut of 7½ ft. on a seam 6 to 8 ft. in thickness, averaged, with scraper loading, 180 tons per shift and 0.444 man-hour per ton. Scoops used in scraper loading range from 1 to 5 tons in capacity.

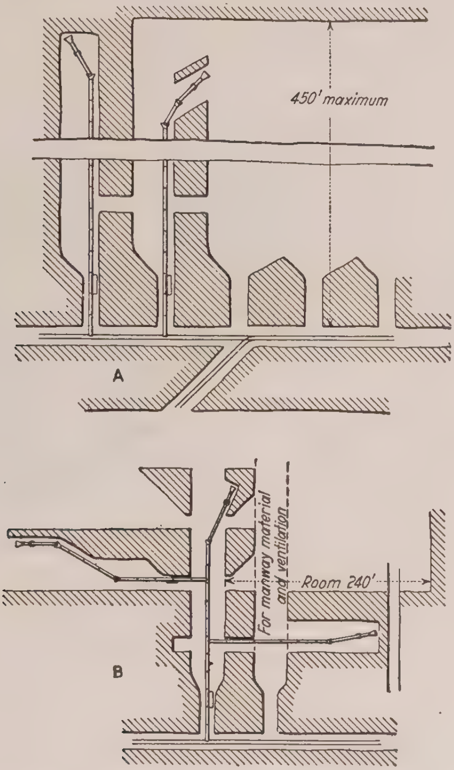


FIG. 295.—Shaking-conveyor units; *A*, rooms extended up the dip; *B*, rooms extended parallel to the strike. (*O. G. Sharrer.*)

The shaking conveyor with duckbill loader is widely used for entry driving, room working, and pillar recovery. In the mines of the Union Pacific Coal Co. 2½ cuts, 15 ft., are averaged in entry driving. Pryde states that rooms 300 ft. in length, which formerly required 1½ years for driving and pillar recovery, are now driven and the pillar withdrawn in 3 to 5 weeks, this result being attained by double shifts and the cutting of room faces and pillars several times in each 8-hr. shift. No outstanding difficulties were experienced in driving rooms under frail roofs. Layouts for room-and-pillar working with shaking conveyor and loader are given in Fig. 295; *a* indicates the driving of rooms up the dip and *b* the driving of rooms parallel with the strike. In both cases pillars are recovered by retreating to the entry, the successive cuts being protected by small pillars next to the caved ground and supported by props. The completion of a pillar section is followed by caving of the top. Rooms are turned on 45-ft. centers and pillars are 23 ft. in width. They are supported by 10-in. crossbar, each placed upon five posts, the bars being used with 6-in. lagging and spaced at 4-ft. intervals. Two 25-hp. shaking conveyors with duckbills, two mining machines, two coal drills, and two blower fans are used. The crew consists of two machine men who cut, drill, and blast the coal, one duckbill runner, and two helpers, or a total of five men. The foregoing example and the figures are taken from a description by O. G. Sharrer.

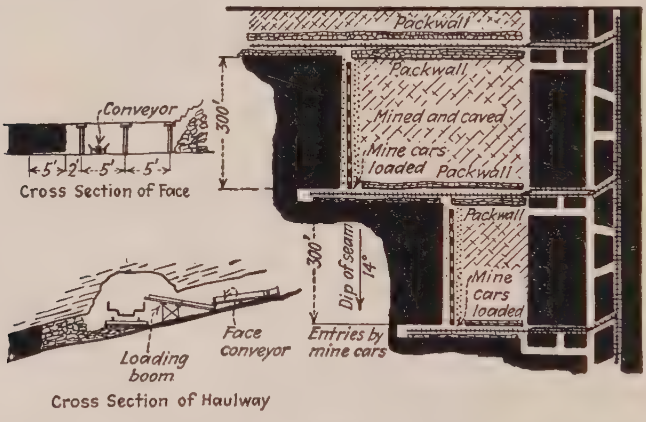


FIG. 296.—Long-face advancing system with conveyors. (*G. B. Southward.*)

An example of long-face mining with conveyor equipment is taken from a description by G. B. Southward. Figure 296 shows the plan of the workings. The coal seam is from 2 to 3 ft. in thickness, with a dip of 14 deg. The face served by the conveyor is 300 ft. in length and is undercut 5 ft. at each cycle. The conveyor is laid about 4 ft. from the face and is moved forward after each cut has been loaded out. A row of timbers is set between the conveyor and the coal. A single entry at the end of each face is kept about 100 ft. in advance of the face to provide loading room for a trip of mine cars. Pack walls support the entry. A short conveyor receives the coal from the face conveyor and delivers it at car height. One face cut, producing from 100 to 200 tons, is loaded out during a shift. Three rows of timbers support the roof overhang, the first row being 2 ft. from the coal face and the third 12 ft. After a face is loaded out, the new row of timbers is placed and the last line is removed, in some cases being blasted out. The roof usually falls shortly after the line of props has been removed but sometimes hangs up for several cuts.

Figure 297, taken from descriptions by Southward, represents an example of scraper loading. Here the coal seam is 4 to 5 ft. in thickness and nearly flat. Panels 270 ft. in width and 1,500 ft. in length are developed by a haulageway and two air courses. Pairs of cross entries driven through the panel width at 120-ft. intervals divide the coal into blocks 120 ft. wide by 270 ft. long. The blocks and the chain pillars are mined by long faces turned off the cross entries on an angle so as to form a V-shaped area in the solid coal. A pair of faces on one entry mines the chain pillar and slabs a 30-ft. width from the 120-ft. pillar, leaving a block 90 ft. wide. This is worked by a single face 120 ft. long turned off of the next cross entry ahead. Only one scraper works at a time in a panel, either on a single face or on double faces that start at the air course and mine down to the haulageway before the next block is started. The scraper, 1,200 lb. in capacity, is handled by main and tail ropes on a three-drum hoist, the third drum of which is used to shift the position of one of the sheaves so as to change the direction of the scraper travel when the scraper is turning from the face down into the entry. The coal is undercut about 6 ft. The roof is supported in the V-shaped area between the faces and caved in the area behind the end of the faces. A row of timbers is set 6 ft. from the face and parallel to it after cleaning up each cut; these are split posts and are set on 4-ft. centers. No timbers are recovered. Usually the end of a face marks the roof break. Loading and timbering are done on the day shift; cutting and drilling are done on the night shift. The day crew consists of the face boss, a hand shoveler, a timberman, a hoist operator, and a car trimmer; the night crew consists of two machine cutters and one helper; a gathering locomotive and two men deliver cars but also serve other work. Usually a face 120 ft. long or a

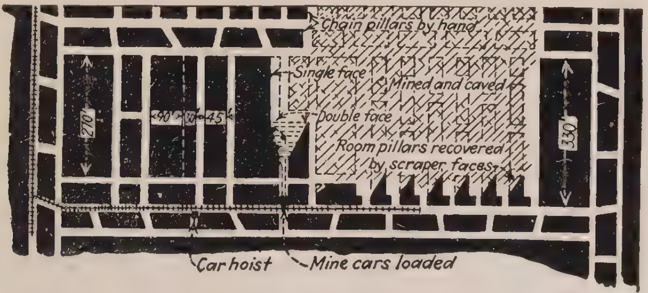


FIG. 297.—Scraper transportation; V faces in rooms and pillars. (*G. B. Southward.*)

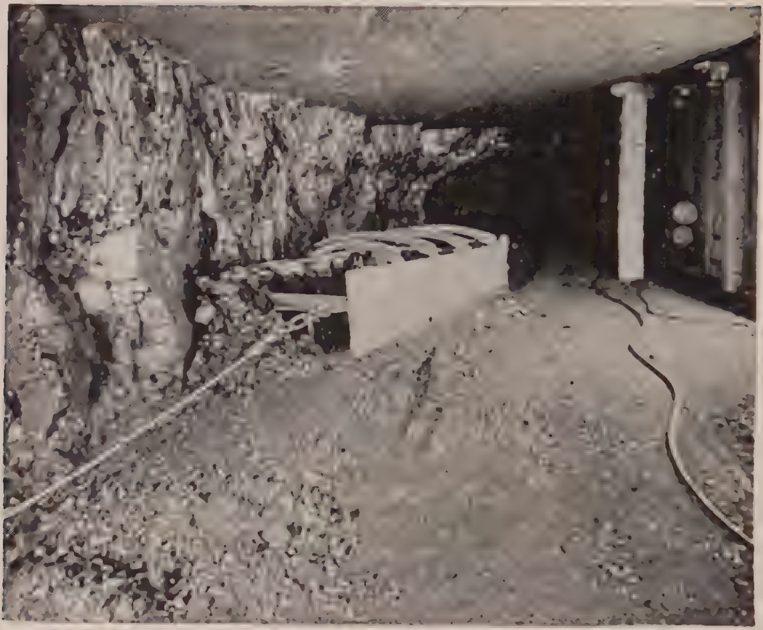


Fig. 298.—Scraper operation in long-face mining of coal.

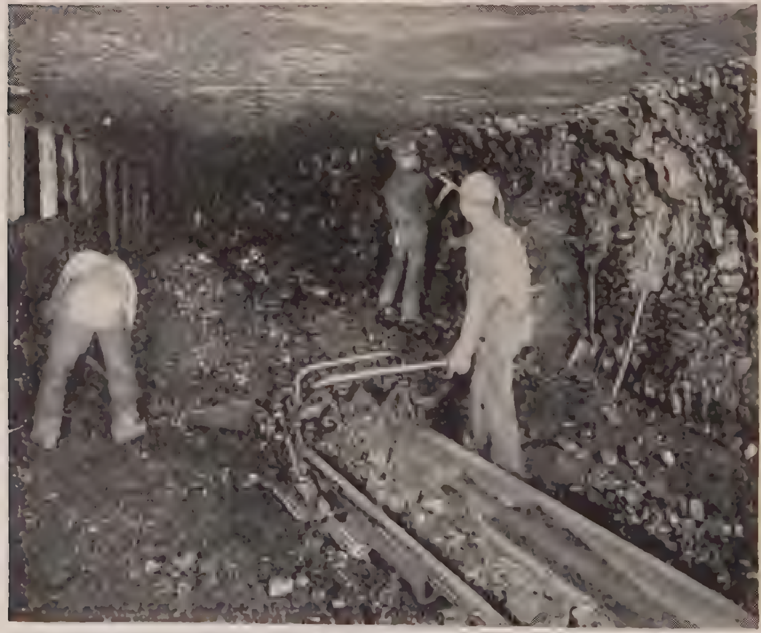


Fig. 299.—Shaker conveyor with duckbill loader.

pair of double faces is loaded out each day, making about 100 tons per day.

Figure 298 shows scraper operation in long-face mining; Fig. 299, a shaking conveyor with duckbill extension; Fig. 300, loading coal at entry; and Fig. 301, a view of caved roof rock. All photos show the mining practice of the Union Pacific Coal Co.

There were 11,810 coal-cutting machines in operation in 1938 in underground bituminous coal mines. Undercut coal was 87.5 per cent of total, the average output per cutter being 23,566 tons. Of a total of 122,581,133 tons, 90.9 per cent of the coal was drilled by electric drills and 9.1 by pneumatic drills. Total drilling machines numbered 6,536; 5,071 were electrically operated and 1,465 were air-operated, with some 851 such drills used on rock work. Power-drilled coal accounted for 38.5 per cent of total underground coal, while mechanically loaded coal accounted for 26.7 per cent. In 1938 there were 1,405 mobile loaders, 117 scrapers, and 1,872 conveyors in bituminous mines handling 85,092,836 net tons. Of underground coal a total of 318,137,909 net tons was divided into hand-loaded, 233,045,073 net tons, and machine-loaded, 85,092,836 net tons. In the same year 10,151,669 tons of Pennsylvania anthracite were mechanically loaded and some 27,990,000 tons hand-loaded.

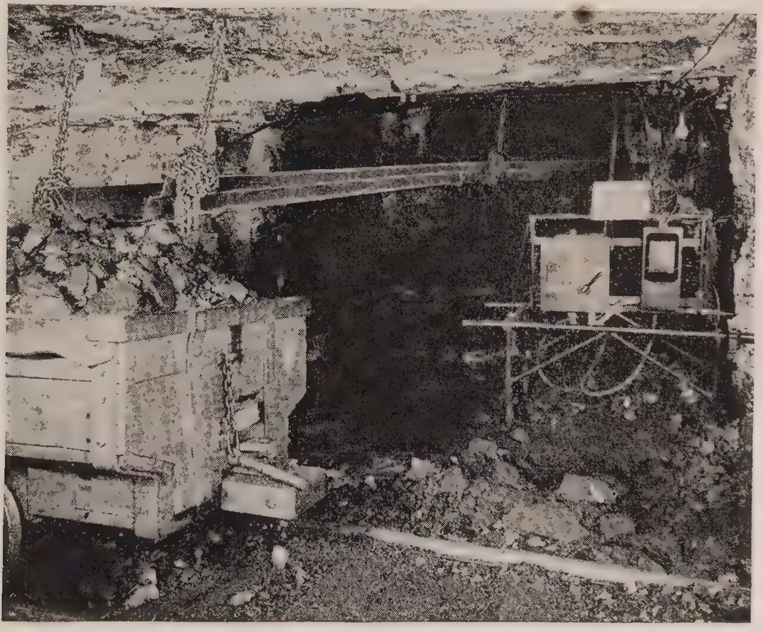


FIG. 300.—Shaking conveyor loading coal at entry.

Mining Salt.—Two methods involving different features from any described are sometimes used. Where water is present in the salt measure, boreholes may be sunk and the brine pumped. If water is not present it may be introduced into the borehole, the salt dissolved, and the resulting brine pumped out. In order to accomplish this the borehole is cased and an air-lift tube is inserted. The outer casing terminates at the top of the salt measure while the borehole is continued to the bottom. The lift tube extends to the bottom of the bore. Water introduced between the inner and outer pipes dissolves the salt. The increased density of the resulting salt solution causes it to sink to the bottom of the borehole, from which point it is elevated by the air lift. In underground mining of salt, sprays of water may be directed against the salt and the resulting salt solutions pumped to the surface.

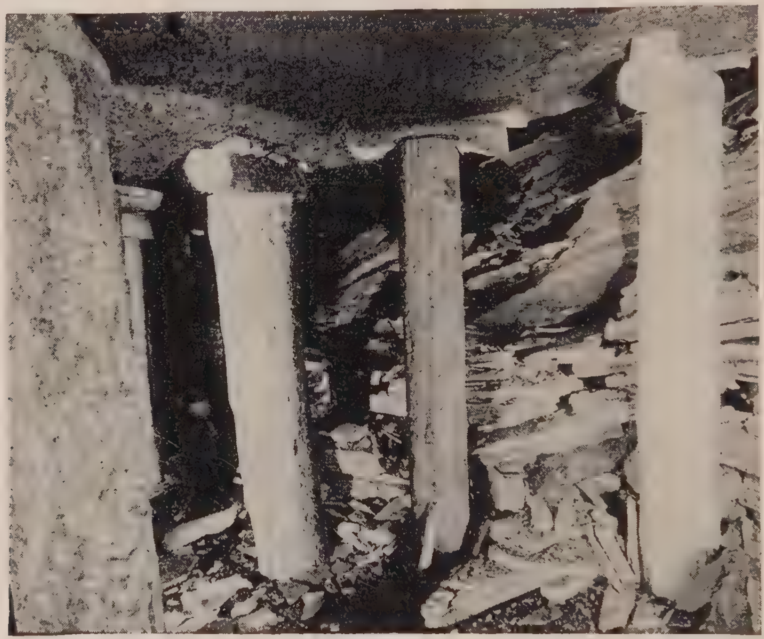


Fig. 301.—Caved roof rock and roof breaks at a line of props.

Mining Sulphur. *Frasch Method.*—This method is employed in the sulphur deposits of Louisiana. The sulphur deposits are covered by clay, sand, and gravel and rest on limestone at depths of 500 ft. or more. A borehole is put down and within each bore four lines of pipe, 10, 6, 3, and 1 in. in diameter, are placed. Superheated water (335°F.) is pumped down in the space between the 6- and 3-in. pipes. This melts the sulphur, which gravitates to the bottom of the bore, from which point it is lifted by the air-lift pump (the 3-in. and 1-in. pipes) to the surface. The molten sulphur is run into large rectangular vats (250 ft. wide, 350 ft. long, and 40 ft. high).

Production of crude sulphur in 1940 was 2,732,988 long tons; shipments were 2,558,742, valued at $40,900,000. Texas produced 81 per cent of the total from five operations. Mining sulphur in Texas followed the discovery of large sulphur deposits in 1918. Known reserves of sulphur continue to expand.

Underground Leaching.—The possibility of removing the metal content of an ore by leaching in place has attracted both miners and metallurgists. Some success has attended a few efforts of this kind, and a few failures have indicated that only a most favorable set of conditions will ensure success in this kind of mining. These conditions are the general permeability of the ore mass through the presence of numerous cracks and fissures, the solubility of the ore mineral in water or dilute sulphuric acid solutions, the impossibility of spreading of solutions to offsetting channels, and the practicability of collecting the solutions in workings below the ore bodies and conducting them without loss to precipitating troughs where the metal can be readily precipitated and gathered. Copper ores have been treated in this way at Bingham and at Cananea.[1] At Bisbee low-grade ore dumps as well as waste-rock dumps and some underground ore have been successfully leached and the copper has been recovered.

Mining Nonmetallic Minerals.—In general, the mining of nonmetallic minerals is by methods similar to those employed in the mining of metals and coal. Wherever possible they are mined by open-cut methods, but in numerous instances the deposits are at such depths as to make underground methods necessary. Salt, gypsum, limestone, silica sand, potash, phosphate rock, and kernite are won by single-entry development and room-and-pillar mining. From 75 to 80 per cent of the mineral bed is mined and from 20 to 25 per cent is left in place for the support of the workings. Clay, diatomite, and phosphate rock are sometimes mined by top slicing, by which method all the mineral is obtained. Talc, fluorspar, barite, feldspar, and magnesite are mined in open stopes, pillar-supported where the beds are inclined at low to moderate angles; where steeply inclined they are mined by shrinkage stoping. It is seldom that waste-rock filling is employed. About 20 to 25 per cent of the mineral is left in place as pillars for support.

GENERAL FEATURES OF MINING METHODS

Stope Limits.—Stope limits are established either by definite walls where these exist, or by the lowering of the grade of the ore. There may be a well-marked lessening in the visible mineral content or a gradual diminishing of values that cannot be discerned by the eye. In the latter case the stope limit must be established by sampling. Short cuts can be driven in advance of the face and sampled or holes can be drilled and the drillings assayed. In gold mines where the gold is free the drillings are panned. The end limits of the stope along the strike of a vein are determined by sampling the drifts. Within the stope the end faces are also sampled unless the ore clearly indicates to the eye its passage into waste. Good mining requires that not a pound of profitable ore be left in the stope, and only close inspection and frequent sampling will attain this objective.

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[1] Underground Leaching at Cananea. C. C. Greenwood, *Eng. Min. Jour.*, vol. 121, p. 518; see also Leaching at Ohio Copper Mine. F. E. Wormser, *Eng. Min. Jour.*, vol. 116, p. 665.

Measurement of Stope.—In regular workings the measurement of the outlines of a stope can often be readily made. A stope book in which the different floors of the stope are shown in plan is used to record the measurements, which are taken at regular intervals, either each week or month. In irregular workings the output of the stope can be measured by the number of cars of ore produced each shift. A tally board is often placed at each chute. All chutes are numbered, and the record is made as the cars are loaded. Surveying methods used in measuring the volume of underground workings are described in treatises on mine surveying.

Testing Loose Rock and Walls.—In overhand stopes it is particularly necessary to guard against falls of rock from the back. After each blast the back should be carefully barred down and all loose rock removed. Long bars are used so that the miner need not endanger himself. After the back is barred down it should be carefully examined and sounded with a hammer where possible. If it is weak, timbers must be placed before work is resumed. Walls are also carefully examined, and any loose slabs are timbered or removed.

Comparative Safety.—There is a certain hazard attendant upon all methods and it is difficult to say that one method is more hazardous than another. Under certain conditions a method may be particularly suitable and its employment will result in comparatively few accidents, while the same method under the same conditions but with inexperienced miners may appear to be extremely dangerous. Probably the most critical period is during the beginning of operations. As experience is gained and as the limitations are better understood the danger reaches a minimum. The employment of miners familiar with a given method reduces the risk to a considerable extent. Poor supervision and inspection and lack of discipline are often causes of more accidents than peculiarity of method. Rapid change in the physical nature of the deposit and the enclosing rock may cause a dangerous situation before it is fully realized. Experience, good discipline, good supervision, and frequent examination of the physical characteristics of the deposit as it is opened up will avert a contingency of this nature. A method obviously dangerous as applied in a given mine should be discarded. A careful consideration of the physical nature of the ore body and of the methods of support required is essential as a preliminary to the selection of a method.

TABLE 120

Method	Per Cent Won
Underhand stoping	95–100
Underhand stoping with pillars	70–80
Overhand, open stope	95–100
Overhand, waste-filled stope	90–95 variable.
Square-set stoping	92–100 variable.
Shrinkage stoping, small ore bodies	90–100
Shrinkage stoping without pillar recovery	70–80
Top slicing, Mesabi	95-98
Bingham	90–95
Top slicing and caving	80–95
Top slicing and shrinkage stoping	70–85
Block-caving	80–90
Longwall (coal)	95–98
Room-and-pillar	60–90
Bord-and-pillar	90–95

Proportion of Mineral or Ore Won.—There is loss of mineral in every method, but the loss is reduced to a minimum where the method is systematically applied and there is close supervision. The approximate proportion of mineral won by different methods is given in Table 120. The proportion of coal won in working coal seams by the room-and-pillar method is given for 12 coal mines by H. N. Eavenson below.

TABLE 121

	Area developed, per cent		Area worked out, per cent
By headings	32.4	By headings	32.4
In rooms	16.4	Chain pillars	2.9
In room pillars	28.1	Rooms	36.7
In barrier pillars	23.1	Room pillars	28.0

Factors That Influence Cost.—The most important factors that influence the cost of mining are method of development, method of mining, size of the deposit, hardness of the rock, presence or absence of water, method of drilling, number of free faces, support, ore handling, continuity of work, wages, hours of labor, experience of the workers, supervision, and mechanical equipment. Of somewhat less importance are plenty of sharp tools, good lighting, and ventilation.

Other things being equal, the larger the deposit the more economically it can be worked since nothing that will reduce cost need be absent from the mechanical equipment. Power haulage, power drilling, lighting, and ventilation can be adequately provided for. Ore in wide stopes can be more economically broken than in narrow. Wide stopes in heavy ground are more difficult to support than narrower ore bodies and are therefore somewhat more expensive to work. The harder the rock, the more costly it is to break, but on the other hand the cost of support is less. Experienced workers under a good working organization and supervision can produce ore at a lower cost than under opposite conditions.

The influence of the method of development corresponding to different methods of mining has been discussed in the chapter on Development (development ratios), and in the division that follows the more important items of expense are compared for different mining methods.

Methods requiring the breaking of all the ore by drilling and blasting are obviously more costly than those in which breaking is accomplished by caving or ground pressure. Ore-caving methods have the advantage in cost of breaking over other methods. The introduction of the stoper and one-man drill has materially diminished the cost difference between the two groups. Breaking in stopes, even where the rock is hard, can probably be accomplished at 15 to 40 cents per ton.

Methods that require support by timbers and timber and filling are more costly than those where timber is dispensed with. The latter methods can, however, be used only where the ore is solid enough to support itself across the width of the working. Block-caving has the advantage of being applicable where the ore is too weak to remain unsupported in a stope. Methods requiring both timber and filling are on the whole more expensive than those where timber alone is necessary. Pillar support has the disadvantage of tying up a portion of the mineral indefinitely; and while there is no money expenditure involved, the profit that might accrue from the working of pillars is lost. Shrinkage stoping where it can be applied has a decided advantage in lower cost over the timbered stope.

Ore movement is an important element in the cost. Methods that eliminate ore handling in the stope and permit chute loading of the cars have the advantage in lower cost. Shrinkage stoping and block caving are conspicuous in this respect. Most of the overhand stoping methods admit of chute loading on the levels, but more or less ore handling in the stope is necessary.

Block-caving is one of the low-cost mining methods. Next in order may be placed shrinkage stoping combined with block or pillar caving, and then shrinkage stoping. Probably the most expensive method is the square set and fill. For thin coal seams the longwall method is more economical than room-and-pillar, and for thicker seams it is an open question which has the advantage. For very thick seams the room-and-pillar or bord-and-pillar can be more advantageously used.

Selection of Mining Methods.—Many classifications of mining methods have been proposed in an effort to find a logical arrangement to cover the variations in mining practice. There are five basic methods of underground mining: Stoping, shrinkage, sublevel, top slicing, and miscellaneous caving and block-caving; in mining bedded deposits longwall, room-and-pillar, and to some extent top slicing are the primary methods. They may be applied as advancing or retreating. In advancing, mining is begun when contact is made with the mineral; in retreating, only after the outside limits of the mineral have been reached, the mining retreating toward the initial entrances with abandonment of the workings as soon as they have been mined out. Mined-out workings may be left open, timbered as they are worked, filled with waste rock, timbered and filled, or allowed to cave.

Stoping may be applied to many types of ore bodies, all having characteristically some self-sustaining properties, ranging up to strong, hard ore masses; comparatively weak masses with strong support may also be mined. Shrinkage is applied to steeply inclined ore bodies where ore widths are 8 ft. or more and where walls and backs are reasonably self-sustaining. Sublevel stoping is applied to medium to strong ore bodies. Top slicing especially applies to soft, weak ores under weak overburden but is sometimes applied to medium-strong ore bodies. Block-caving is applicable to large, thick ore bodies of low to medium strength but in some cases may be applied to relatively strong masses such as limestone and asbestos rock.

In any one mine a variety of methods may be applied, each designed to meet the conditions of the individual ore body. Safe operation is the first consideration. Next is the cost question. Superintendents, foremen, and engineers determine the applicability of the one or more methods that may be considered. Experience is a potent factor in the selection of a method.

Selective mining, sorting in stopes, and ore grading are restricted to stoping, top slicing, and sublevel caving methods. Others take the whole ore mass. Where it is necessary to mine several ore grades, as in some iron mines, bulk mining methods are avoided. The present trend is toward selective mining, as labor costs, taxes, and other expenses have increased. These considerations may be a factor of importance in the selection of a method.

Caving methods introduce the probabilities of surface subsidence, and consideration of its extent is necessary. The stability of shafts and surface structures may be affected adversely. Property damage likewise needs consideration.

Stoping Costs.—Narrow ore bodies cost more to mine than wide ones; below 10 ft. in width costs increase much more rapidly than they decrease as the width increases. Small dabs of ore cost more than large ore masses. The wider the stope, usually the more tons broken per machine shift. Stope widths of 40 to 50 ft. may give 40 tons or more broken per machine shift; 25 ft. in width, some 25 to 30 tons; at the Nevada-Massachusetts shrinkage stopes of 4.5 ft. in width gave 11.4 tons per man-shift. At the Old Eureka mine (California) underground production was at the rate of 1.25 tons per man-shift.

Hard ores cost more to drill and blast. Heavy timbering delays mining and costs are increased. Poorly drained stopes increase costs. As conditions vary widely from mine to mine, comparative costs are almost useless unless such conditions are accurately known. Useful tables of stoping costs are to be found in *Bulletin* 419 of the U.S. Bureau of Mines.

Elsing gives examples of open stoping that average 60 cents per ton and with development, transportation, and other items reach 90 cents per ton. Cut-and-fill stoping shows a cost range of $2.53 to $4.51 per ton; square set, $2 to $4 per ton; sublevel, 40 cents to $1.50 per ton; sublevel caving, 60 cents per ton; top slicing, 75 cents to $1 per ton; and block-caving, 25 to 50 cents per ton. At the San Francisco mines (Mexico), shrinkage stoping cost $1.18, bench stoping $0.98, and cut and fill $1.65 per ton (1936). At Ymir in 1937 narrow ore bodies cost $0.76 for development, stoping by shrinkage $1.39; total mining costs were $4.62.

Table 122 gives details of comparative costs at one mine for various methods and conditions. Present-day costs would be greater than these. In Table 122*a* incomplete costs are given for the United Verde mine for several mining methods (1940).

Mining by block-caving gives from 13.2 to 31 tons per man-shift with explosives ranging from 0.08 to 0.6 lb. per ton., power from 2.47 to 4.75 kw.-hr. per ton, timber 1 to 1.75 bd. ft. per ton, and concrete where used 0.03 to 0.05 cu. ft. of cement per ton. Actual mining costs per ton are not available but are usually lower than by many other methods.

Alaska Juneau reported in 1937 mine operating costs of 34.4 cents per ton with total operating costs of 65.5 cents per ton; for the period 1914–37 inclusive, mine operating costs for 64,976,504 tons trammed were 28.1 cents per ton and total costs were 63.5 cents per ton for an ore value of 87.2 cents per ton and a net operating profit of 23.7 cents per ton.

In 1940 Transvaal gold mines milled 64,515,350 tons, giving an average yield of 4.196 dwt. per ton; total working costs were £1 0*s*. 8*d*. per ton of ore, and £4 18*s*. 6*d*. per ounce of gold; working profit was 14*s*. 9*d*. per ton.

An idea of the cost of mining coal can be obtained from a consideration of the average price realization per ton. The following figures are for 1929. The total of independent companies was 4,386, operating 5,570 mines, and realization was an average of $1.72 per ton; average per man

Table 122a.—Stoping Costs at United Verde Mine, 1940

(*J. B. Pullen*)

	Remnant stopes	Vertical pillar	Horizontal pillar	Mitchell slice	Incline cut and fill
Tons mined	7,239	7,100	9,260	91,157	177,019
Cost per ton:					
Labor	$0.537	$0.578	$0.624	$0.589	$ 0.462
Timber	0.235	0.184	0.287	0.224	0.128
Explosives	0.078	0.061	0.160	0.090	0.136
Total cost	$ 0.850	$0.823	$1.071	$0.903	$ 0.726
Lb. explosive per ton	0.44	0.35	0.93	0.52	0.78
Bd. ft. per ton	7.47	5.89	9.26	7.18	4.08
Tons per man-shift	10.46	9.06	8.74	9.35	11.87

Note.—Power not included.

Table 122.—Comparative Stoping Costs,[a] Calumet and Arizona Mine[d]

Method and conditions	Labor	Explosives	Timber	Candles or carbide	Air	Other supplies	Total
Square-set, oxide ore,[b] heavy ground	$0.73	$0.06	$0.34	$0.01	$0.12	$0.04	$1.30
Square-set, oxide ore, average ground	0.60	0.05	0.25	0.01	0.10	0.04	1.05
Square-set, oxide ore, robbing timbers	0.61	0.05	0.16	0.01	0.10	0.04	0.97
Top-slice caving, old method, oxide ore, heavy ground	0.63	0.07	0.20	0.01	0.10	0.03	1.04
Mitchell top-slice caving, oxide ore, heavy ground	0.54	0.04	0.21	0.01	0.10	0.03	0.93
Mitchell slicing, oxide ore, average ground	0.51	0.05	0.20	0.01	0.06	0.02	0.85
Square-set, sulphide ore,[c] average ground	0.49	0.04	0.19	0.01	0.04	0.03	0.80
Mitchell slicing, sulphide ore, average ground	0.43	0.03	0.15	0.01	0.03	0.01	0.66
Cut-and-fill, Gilman method, sulphide ore, good ground	0.34	0.04	0.07	0.01	0.04	0.01	0.51
Cut-and-fill, old method, sulphide ore, good ground	0.32	0.04	0.04	0.01	0.04	0.01	0.46

[a] Figured on the basis of the wet tons mined These costs obtained under normal conditions with a base wage of $4 per day for miners and $3.75 for muckers. All costs should be increased proportionately for present-day conditions.

[b] Oxide ore in place will average from 12 to 16 cu. ft. to the ton.

[c] Sulphide ore will average from 9 to 10 cu. ft. to the ton.

[d] Table taken from article by P. D. Wilson.

output per day was 4.77 tons and average days worked was 214. The total of consumer-owned companies was 226, operating 487 mines, and realization was $1.99 per ton; average per man output per day was 5.13 tons and average days worked was 240. Total companies for the year were 4,612, operating 6,057 mines, and realization was $1.78 per ton; average tons per man output per day was 4.85 and the number of days work averaged 219. The foregoing applies to bituminous coal. In the same year there were 297 mines producing anthracite, average product price $5.85, average days worked 225 and net tons per man-day 2.16. In 1939 bituminous coal companies totaled 5,138, mines 5,820; average product price $1.85, operating days 178, production about 5 tons per man-day, while anthracite mines numbered 363, average product price $3.85, days operation 183, per man-day production 3.02 tons. In 1939 there were 500 bituminous strip-pits in operation, average price $1.53 per ton, days operation 197, per man-day production 14.04 tons.

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CHAPTER XVI

MINE ORGANIZATION AND OPERATION

Organization.—Some mines grow piecemeal, starting on a small scale. The first proceeds are used to build a small mill. The development work is extended and more ore is discovered. Perhaps an air compressor and hoist are then added, and the nucleus of a mining plant is formed. The enlargement of the mill follows. Thus bit by bit the facilities for working are increased. The mine, known locally as a "good little mine," begins to attract outside attention. The owners extend development, and the amount of ore discovered shows the deposit to be of importance. The owners may be content to work along in this manner, but in many cases they are tempted to sell out and let someone else bear the burden of large-scale working. If they are wise they bring their mine to the attention of some strong company or individual operator who makes a business of buying and operating mines. They employ an engineer to prepare a report and submit this to the representatives of the company. If it is favorably considered the company sends engineers to make an examination, and acting on their report they reject, buy outright, or secure a "working bond." The working bond gives the privilege of working the mine for a period of a year or more with the option of purchase for an agreed-upon sum at the end of the period. Usually one or more partial payments upon the purchase price must be made at the beginning and during the life of the bond. Vigorous development is the keynote of the initial operations of the company. They endeavor to prove the possibilities of the deposit as rapidly as possible. If they are successful a new company may be organized, the mine is equipped, and the production of ore is started. While some mines are operated by individuals a company organization is customary.

Capital, both for the purchase of the property and for equipment and operation, is secured from the public or private sale of shares of stock in the company.

Company Organization.—A company organization includes the president, the board of directors, the secretary, the treasurer, the auditor, and the stockholders. The company is organized under a charter, which is granted by a state or other governmental authority. In principle the charter may be likened to a constitution. It defines the nature of the business, the capitalization, and other general features.

The bylaws of the company specifically define the authority and duties of the company officers, prescribe the method of paying dividends, the disposal of stock, the amount of debt that may be incurred, the depositing of money in banks, and the manner of its withdrawal. They are so drawn as to leave nothing to personal discretion.

The president is the chief executive authority. He is responsible directly to the board of directors and through them to the stockholders. He is the business head of the organization. The directors are the direct representatives of the stockholders. They serve as a check upon the president. Any proposed action must meet with their approval before it can be carried out. The secretary records the minutes of meetings, receives and transmits communications to the proper officials, and transfers stock. The treasurer is the custodian of the funds. The auditor checks the expenditure of all moneys and the accounts of the company.

The stockholders are the investors in the company. It is their money that is used to purchase, develop, and operate the property. They elect the board of directors, adopt and change bylaws, and in some cases sanction the exchange of stock for property. A majority of the stock shares controls in elections and other business that may be submitted at a stockholders' meeting. Stockholders' meetings are usually called annually or when necessary by the president under the conditions named in the bylaws. Directors may meet monthly or when called by the president.

The financial organization is upon either the assessment or the treasury-stock plan. In the former, when the company funds are exhausted, an assessment is levied and collected for each share of stock. In the latter, a part of the total number of shares is retained as treasury stock. This stock is sold as occasion for additional capital arises.

In the distribution of the earnings the practice in well-managed companies is to accumulate a surplus, which is reserved for operation. All earnings in excess are distributed in monthly, quarterly, or semiannual dividends. The surplus, when the mine reaches a worked-out condition, is used either for further prospecting or for the purchase of a new property, or it may be returned to the stockholders.

The chief operating official is the general manager, sometimes called the managing director, general superintendent, or superintendent. He is selected by the president and board of directors. Whether the mine is small or large the individual selected for the direct charge of the property must have technical knowledge and experience and must have shown ability to manage men. Personality and character are not overlooked. Good management is one of the first requisites toward the success of a mining enterprise, and a man who has a successful record inspires confidence in the minds of the stockholders and directors. Tact, a keen business sense, and balanced judgment are essential factors in the success of a manager.

The general manager selects his own staff of technical assistants. As the members of the staff are directly responsible to the manager, it is desirable that they owe their appointments to him. The staff of a large mine consists of a mining engineer, geologist, metallurgist, mine surveyor, assayer, mechanical and electrical engineer, accountant, and very often a physician. The members of the staff are directly in charge of the separate departments or divisions of the work. The accompanying chart illustrates the complete organization (Fig. 301*a*).

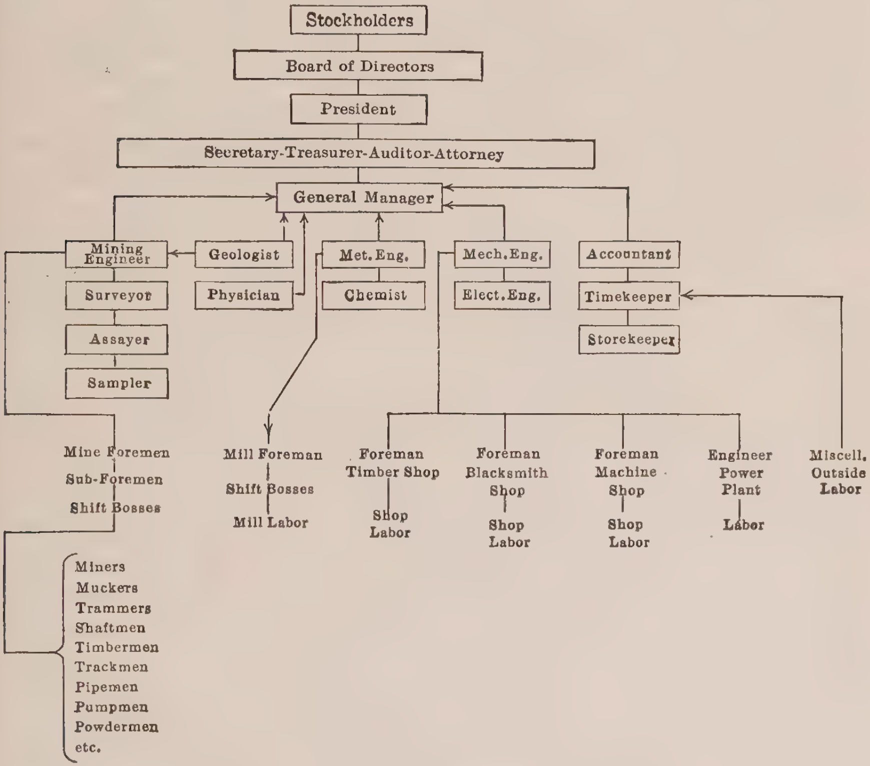

FIG. 301*a*.—Mine organization chart.

In a small mine staff positions are often consolidated. The manager may assume direct charge of mining operations in addition to his managerial duties. The shops may be under the general charge of a master mechanic. The mine surveyor may in addition sample ore and record geology. The accountant may act as timekeeper and storekeeper. The assayer makes all of the assays and analyses for both mine and mill and may in addition be in charge of the ore-treatment plant. There are many examples of successful mine operation where the staff positions are few in number.

Principles of Management.—In the operation of a mine, labor, power, materials, and mechanical appliances are brought together to accomplish a specific end, the winning of ore or mineral, its treatment, and the marketing of the products. Profit is the dominating motive. Stockholders put their money into an enterprise in order to make more money. The success of the business is measured by the dividends returned. In order to pay dividends the income must be greater than the outgo. Income is limited by the grade of the ore, the percentage extracted, and the selling price of the product. Outgo is controlled by good management. Good management means the close control of expenditures, efficient working, and the coordination of all the parts that go to make up the whole. A comprehensive plan, a well-designed plant, and the careful selection of staff men, foremen, and workers are essentials.

The human element is the important directive feature of any business organization, and upon its proper control depends much of the success of the management. A supreme head is necessary, and this not only applies to the position of general manager but to each department head as well. Each department head is responsible to the general manager, but within the department he is the final authority. The same principle applies to the foreman or mine captain. He is given complete charge of the shift bosses and workers and often authority to engage and discharge. In large mining companies, the employment manager assumes the task of engaging and discharging men, removing this responsibility from foremen and mine captains. This plan has been found to be more equitable to the men employed and is preferable to the older method. In small operations, the superintendent may assume the role of employment manager with advantage to everyone employed. The latter power gives him control of the situation besides making him responsible for the selection of efficient men. The limitations of each directive and staff position should be clearly and definitely stated and should reasonably conform to the abilities of the men holding them.

The burden of the administrative work should be equitably divided. Each man should have sufficient to occupy his full time, but he should not be overloaded. Underloading is probably more objectionable than overloading and a manager must be closely in contact with his men in order to make any necessary adjustment in the division of the work.

Close application, good work, valuable suggestions, or extra service should be rewarded by promotion or a salary increase or at least by commendation where a more substantial recognition is impossible. Loyalty to an organization on the part of the individual arises from sympathetic understanding and a feeling that an advance in position or salary is something more than a daydream. In well-managed organizations a definite succession of positions is established, and men are selected for advancement on account of either long service or exceptional ability. The opportunities for promotion in small organizations are often few and far between, and under such conditions a manager can prevent his organization from getting stale by assisting the more ambitious men in securing positions of larger responsibility in other mines. Staff men fall into three types: Efficient ambitious workers who are seeking to advance themselves, contented efficient workers who may be termed the wheel horses of an organization, and the younger men who are securing experience and are eager to be tried out. Men of the first type when they reach a dead end often become discontented, and sometimes discouraged and inefficient. Such men constitute a problem to the manager.

A system of daily, weekly, and monthly reports, summarizing the activities of each division of the work, should be initiated. These reports should not be too voluminous but should be sufficient to keep the manager in touch with each department. Each department should have a system of supplementary reports that would serve a like purpose for the department head.

A filing system in which correspondence, reports, contracts, plans, drawings, and specifications relating to the company's operations are systematically arranged is necessary. Such a file serves as a record of all operations and enables the manager to refer to any past detail without loss of time. In a similar manner, though perhaps on a smaller scale, each department maintains a file that serves as a record of the detail of the department.

The system of accounts should be carefully planned by the accountant and reviewed by the manager. The books of account serve as a record of all financial transactions.

An inventory of company property and equipment is also maintained. Where changes in staff positions are made the individuals concerned become responsible for the proper checking of the department equipment. Inventories should be checked at regular intervals.

All operations should be analyzed and carefully studied with a view to simplification and the elimination of unnecessary steps. For this purpose sequence charts and time analyses are made and the results of changes are reviewed. The fact that time is required to bring operations into harmonious adjustment should not be overlooked in making changes. Too many radical changes may result in more or less confusion and in the end may defeat their purpose. Conservatism is not without its advantages.

Improvements in methods of mining and ore treatment and in mechanical appliances should be noted and their possible application to the problem in hand given consideration.

Comparisons should be made with the operations of like mines, and their cost accounts and performance should be brought to the attention of the staff members. A broad policy would make it possible for the manager as well as the staff members to visit other mining localities with the object of making detailed studies of similar operations.

Interruptions to the regular operation of the mine and ore-treatment plant should be made as few as possible. Future changes and emergencies should be anticipated and provision made in ample time to prevent any serious delay in operations. A sufficient stock of supplies to provide for unusual delays in shipment should always be carried.

Concentration of the work and of the workers both on the surface and underground is an important principle, the application of which results in the reduction of the equipment underground, the simplification of the work of the foreman, and a reduction in costs.

A follow-up system should be worked out to suit the peculiar requirements of the mine. The purpose of this is to see that orders are promptly executed or that the reasons for delays and nonexecution of orders are given to the individual in charge of the division. This can be done by inspection, conferences, or written reports.

Safe operation and the reduction of the risk element, which is always present, to its least dimensions should be a feature of the operations. Safety appliances and emergency apparatus should be liberally provided. New men should not be allowed to work in dangerous places, and workers should be encouraged to prevent accidents.

Frequent conferences should take place between the manager and his staff as a whole and special features of the company's operations should be thoroughly discussed. Safety and welfare committees should be appointed and encouraged to take up subjects of common interest to both staff and workers.

Not without importance in the general scheme of management is the question of living quarters for staff and workers. In many mines this is not an important matter, but in others living quarters must be provided out of the company funds. A liberal policy in providing adequate quarters and encouraging social and amusement features is compensated by increased efficiency and greater contentment.

Duties of Staff Positions.—The mining engineer or the mine superintendent, as he is more often called, is in immediate charge of all operations concerned with the development and mining of the ore. The plan of development, the method of mining, ore handling and transport, the details of drilling and blasting, the division of mine labor, the timbering, drainage, and ventilation, and the underground prospecting are subjects that he must collectively review and whose details he must decide upon. Chutes, stations, ore pockets, shaft timbers, headframes and ore bins must be designed and constructed. His immediate assistants are the foreman, underforemen, and shift bosses.

The mine surveyor measures the advance made in the development workings, usually each week, plots the extensions on the mine map, determines the limits of the various stopes from time to time, makes volume and quantity estimates, computes the course and distance required for underground connections, and lays out and checks up the courses in the mine. In addition, transverse and longitudinal sections of stopes and ore bodies as well as stope maps are constructed and kept up as the deposit is developed and worked. Where required, surface surveys and topographic maps are made. A mine model showing all the workings is not infrequently maintained and serves a useful purpose in establishing a clearer conception of the ore bodies and the related workings. Where buildings are to be constructed the lines and elevations are determined by the surveyor. A complete record of all survey notes and computations is kept.

In large mines, or in a group of mines operated under one management, the interpretation of the structure and direction of the underground prospecting is placed in charge of a geologist. It is his duty to direct the prospecting work where the extensions of a vein have been lost by faulting, to record the geological facts, and to determine the possibilities of unprospected portions of the property. In small mines a consulting geologist is sometimes engaged temporarily to study the possibilities of the mine and to advise upon the direction of new work.

The sampler, who may work under the direction of the mining engineer, surveyor, or geologist, has the special duty of taking samples in development workings and stopes, recording the position of the samples, and delivering them to the assayer. The sample records are entered upon an assay map, which is usually kept up either by the mine surveyor or the head sampler. The assay map is the record of the position and value of the samples taken. Its principal use is to establish the limits of the stopes and to facilitate the computation of average values and tonnages.

The assayer has charge of all of the determinative work upon mine and ore-treatment samples. He reports to each department concerned and submits a general report to the manager as well.

The metallurgical engineer is in direct charge of all ore-treatment operations. He is responsible for the securing of the maximum extraction and for the details of treatment. The sampling of products and the testing of the chemicals and supplies used are done under his direction.

The mechanical and electrical engineer is in charge of the mechanical plant and repair work upon all of the mechanical appliances used. He is responsible for the maintenance of mechanical efficiency. He must initiate an aggressive policy of thorough and regular inspection, followed up by the execution of repairs as their need is discovered. Unforeseen accidents are prevented by having machinery and appliances in good condition. Certain limits of wear must be established and replacements made when these are reached. The stock of repair parts required for the different machines in use must be maintained in order to minimize delays. He should as far as possible standardize both machines and spare parts.

The accountant is charged with the maintenance of the accounting system. Bills and invoices are received and checked either by the several departments or by the storekeeper. Pay rolls are made up and checks filled out. Monthly reports of outgo and performance and cost sheets of different operations are made. Daily cost reports may be required. The apportioning of general costs of operation is made between the different departments.

The timekeeper is responsible for checking the workers as they report for work and when they leave at the end of the shift. The foremen and shift bosses independently report the number of men at work in each department. Miscellaneous workers about the surface place are checked by the timekeeper.

The storekeeper is responsible for the receipt, storage, and issuance of supplies.[1] Supplies are issued on a requisition signed by the foreman or department head. A card record of the stock on hand and its position in the storehouse is maintained. When a supply reaches a certain minimum an order for replenishment is made. At regular intervals the card record is checked by an inventory. Invoices are checked as to both quantity and price. By means of distinctive requisition blanks the storekeeper can charge individual departments with the supplies used. A debit and credit account should be maintained, each department being charged with the supplies issued and credited with those returned. Supplies are sometimes sold to other mining companies or to miners and prospectors, and transactions of this nature are reported in the monthly statements.

A physician is retained by many companies to provide medical attention for the men in case of accident or sickness. Usually the employees pay a small monthly fee for this service. The physician should be put in charge of all sanitary arrangements about the mine and should have general charge of the health of the men. His advice should be sought in coping with unusual underground conditions affecting the health of the workers. The training of first-aid squads is a part of his routine work.

Duties of Foreman.—All underground labor is engaged by the mine foreman or the mine captain, as he is sometimes called. It is his duty to select and assign the men to the different tasks and to see that the work is properly performed. He must have certain reasonable standards of the amount of work that can be done under the given conditions. The ratio of output or performance to hours of labor expended is his measure of the efficiency of a worker. Where labor is abundant he can weed out the inefficient and incompetent and in time build up an efficient and satisfactory working crew. He must have executive ability of a certain kind and understand human nature. He must be thoroughly familiar with the ordinary run of mining work. There is no one who can more quickly spot an incompetent foreman than the skilled miner. As the success of the underground work rests largely upon the foreman, particular care should be taken in his selection. A wide-awake, practical, and experienced man is needed for the position. The foreman selects his own shift bosses, who are responsible to him. The foreman usually assumes charge of one shift, while the shift bosses are in charge of the others. Both the foreman and shift bosses make at least one round of all the working places during the shift. They check the men at work in the various places, note the nature of the work being done, give specific directions to the workers, sample, and observe any changes in formations. Their report is submitted to the manager each day and keeps him in touch with the underground situation. A timekeeper usually checks the workers as they go "on shift." The timekeeper's report is checked by the foreman's report. At regular intervals the mine superintendent accompanies the foreman on his rounds and thus comes more closely in touch with the operations. The successful foreman must use tact rather than force and must be quick to recognize either merit or soldiering. He should stimulate rivalry between the different working crews. Above all, he should be fair and impartial.

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[1] The stock of supplies carried by a mining company will vary between wide limits and will depend upon the liberality of the management, the transportation facilities, the kind of mining, and local conditions. As an example one large gold-mining company maintains a stock of mine supplies equivalent in value to $0.30 per ton of annual output and a stock of mill supplies equal to about the same figure.

In a large mine, in addition to the foreman and shift bosses, different groups of workers are placed in charge of bosses. These men may have charge of tramming, mucking, track and pipe laying, timbering, shaft work, etc. They receive a somewhat higher wage than the workers and are responsible for the carrying out of the orders of the shift boss. They work with the men.

The shop foremen are responsible for the rapid and economical execution of all repair work. They lay out the work and inspect the finished product. As a rule they work in the shops with their men. Skilled workmen of more or less executive capacity are selected for shop foremen.

Labor.—One or two shifts of 8 to 10 hr. duration is the usual division of the day. The 8-hr. shift is almost universal in mining districts. Contract work, an important feature, is also based upon the 8-hr. shift. In some localities miners are lowered into the mine on their own time and the meal hour is not included, while in others the men are lowered on company time and the shift includes the half hour allowed for eating. The actual number of hours of work ranges from 7 to 8. To make a short shift effective good organization is essential and no time can be lost. In shaft sinking and adit work three 8-hr. shifts are customary.

The working force is divided into surface and underground men. Specific tasks are assigned to each worker. The particular designation of the workers depends upon the mining practice and the locality. In metal mining underground workers are designated as miners, shaftmen, shovelers (muckers), trammers, chutemen, trackmen, pipemen, pumpmen, station tenders, motormen, electricians, etc. Surface men are designated as cage riders, landers, trammers, carpenters, timber framers, blacksmiths, blacksmith's helpers, tool sharpeners, ore pickers, engineers, firemen, etc. In a coal mine the designations are miners, rock miners, timbermen, bratticemen, shot lighters, tracklayers, motormen, motormen's helpers, switchmen, drivers, door boys, rope riders, pushers, cagers, grippers, couplers, etc. Outside men are tipple dumpers, slate pickers, breaker-screen men, boxcar loaders, tally boys, teamsters, blacksmiths, blacksmith's helpers, engineers, mechanics, couplers, oilers, car repairers, timber framers, timber sawyers, etc. Sufficient has been given to indicate the subdivision of the work. The selection and apportioning of the workers is the duty of the foreman and his assistants. It is evident that the synchronizing of the work and the balancing of the number of workmen is a task of no mean magnitude. The working crew at the start of operations is usually small, and the attainment of maximum capacity requires time, sometimes a year or more. There is thus a sufficient interval for the gradual growth of the working organization and its adjustment to the peculiar conditions of a given mine.

Conditions are so varied at different mining properties that it is difficult to obtain comparative data on the number of workmen employed and their respective grouping. A few examples will serve to give some ideas about the subject. At the Hollinger group of mines during 1930 the total number employed was 2,263 for an output of 1,625,863 tons; of these 1,572 were designated as miners, divided into 35 on exploration, 345 on development, and 1,192 on production; 264 mechanics, divided into 102 on operation and 162 on maintenance; 427 general, divided into 187 in mill and refinery, 61 engineering staff, 39 clerical staff, and 140 miscellaneous. Work was distributed over some 29 levels and a number of separate veins were worked. Annual tons per man was 718.

At the Alaska Juneau mine an average of 620 men were employed during 1930 at a total over-all cost of $9.73 per man per day; production was 3,924,460 tons. Annual tons per man was 6,329. The Miami Copper Company employed about 875 men in its mining department and produced over 6,000,000 tons of ore, giving an approximate annual tonnage per man of 7,000 tons. This applies only to mine labor. The Bawdwin mine of the Burma Corporation produced 462,184 tons in 1930–31, the total number of mine shifts being 1,535,716 for the year, the average per day 4,387, and the annual tons per man 105. At the Bisbee mines of the Calumet and Arizona Mining Company the 1930 production was 427,436 tons; the total number of men 728 on part time and the annual tons per man approximately 587. Childs and Easton state that at the Bunker Hill and Sullivan mine (Idaho) 1,175,960 man-hours were required for a production of 421,532 tons, or some 0.358 ton per man-hour; 948,000 man-hours being required underground and 227,960 man-hours upon the surface, a division of 80.6 and 19.4 per cent, respectively. The annual tons per man was 937.

In presenting data on mine operating conditions in the Pocohontas coal field, Edward O'Toole gives the division of labor in percentages of total day labor as follows: Inside labor 42.70; underground transportation 21.14, and outside labor 36.16 per cent; underground labor was 63.84 per cent of the total. Labor efficiency is expressed as follows in tons per man-hour: miners in headings, including machine runners, 2.115; miners in rooms and at pillars, including machine runners, 1.87; miners on rock work 2.5; all miners 1.91; all underground day labor 1.85; men on transportation 5.57; all surface labor, excepting office force, 3.38; and for the total organization, excluding general office and engineers, 0.728.

Day's pay, day's pay on a sliding scale, day's pay and bonus for exceeding an agreed-upon minimum, and payment on the basis of a contract are the methods of compensation. In certain mining localities the day's pay system rules. Either a flat rate for all underground workers and a similar rate for surface workers or a wage scale differing for each class of employment is established. The wage scale is perhaps more satisfactory, as it discriminates between the different classes of work, and jobs requiring greater skill or more arduous work receive a higher wage. Skilled shop labor and engineers usually receive a higher wage than miners.

Day's pay on a sliding scale is in use in the copper mines of Arizona and Montana. A minimum rate is established and when the price of copper exceeds 15 cents per pound the rate is increased for each increase in the price of copper. In principle this method of compensation is an adaptation of the principle of profit sharing.

The bonus system of compensation has as its principal purpose an increase in the rate of working. Unit costs are reduced as well, and in tunnel work the cost per foot for labor is reduced from 15 to 50 per cent. A minimum wage and a minimum footage per day are established. Any increase in the footage above the minimum earns a bonus. The bonus is usually figured as one-half of the labor cost saved on the increase of footage. Adit, tunnel, drifting, raising, crosscutting, and shaft-sinking work can be done in many instances more economically by this system than by day's pay.

Contract work is a feature of most coal mining, and in many metal mines it is used in conjunction with the day's-pay system. The tribute or lease system is perhaps one of the earliest forms of contract working. Under this system the miner extracted ore and received payment by a royalty, consisting of a fixed percentage of the value of the ore. The mining company would often supply timbers, powder, power, and tramming equipment. Under other conditions the mining company would simply lease the block of ground on a royalty basis. The tribute and leasing system is more or less used at the present time. It is common practice in many metal mines to contract drifting, crosscutting, and raising at so much per foot. Tramming and stoping are sometimes contracted for on a tonnage basis. Contract work has the advantage of greater economy, higher rates of working, and the automatic elimination of unskillful workers. On the other hand, skilled workers can earn a greater return than under the day's-pay system. It is probably the most satisfactory form of compensation. The determination of contract rates is based on a thorough analysis of rates of working and labor costs under the day's-pay system. While practice varies in the details of the application of the system it is advisable to have tools, powder, and blasting supplies charged against the contractors, while timber and power are supplied without charge. This arrangement results in greater economy in the use of tools and supplies.

A discussion of the so-called labor question would be out of place in a treatise of this kind, yet it is a vital question to the mine manager and must be reckoned with on the cost sheets. When the management has ensured equitable treatment of the workers, made underground operations as safe as the conditions will permit, provided change quarters and reasonable accommodations for living and eating where such are required, safeguarded the general health of the workers by proper hygienic precautions, and established a club with accompanying amusement features, it has done as much as can be expected.

Efficiency.—Mechanical efficiency is the ratio of work output to work input. Many mechanical appliances are used in mining operations. It is evident that without proper control of the efficiency of the individual machines, power will be wasted and the aggregate waste may be a large and an important item of cost. Mechanical appliances should be selected with special regard to their efficiency as well as to their applicability to the particular work. After installation they should be tested at frequent intervals and necessary repairs and adjustments made as required. Lack of efficiency in a particular machine may be due to improper or irregular lubrication, wear, or want of adjustment. These three factors receive the attention of the mechanical engineer, and his task is never finished while the plant is in operation. Steam lines, compressed-air lines, and electric power cables must be inspected for leakage and kept in repair. Electric apparatus must be protected from overloads and fuses and circuit breakers kept in repair. Workers must be instructed in the proper use and care of mechanical appliances. The objective is the prevention of waste, the maintenance of efficiency, and protection from careless operation.

Labor efficiency is measured in terms of output per hour or per shift. A reasonable rate of working should be expected, and work standards should be determined by careful observation for all classes of work. Comparisons should be made from time to time and under varying conditions. The final measure of labor efficiency appears upon the cost sheet in terms of the labor-unit cost for each division of the work. From the viewpoint of the individual worker, efficiency is attained by the least physical exertion required to accomplish a given task. The most efficient workers are those who accomplish a particular work standard with the least consumption of vital energy.

Staff efficiency signifies the careful planning of the details of each division of the work to the end that a minimum of time, of labor, and of cost is required for execution. The general influence of a well-organized staff is manifest by the maintenance of costs at a comparatively stable figure or decreasing cost from year to year where the conditions are normal.

The efficiency of the management as a whole is determined by the results obtained. The attainment of a maximum output per worker employed, the reduction of unit costs to a minimum, the securing of a maximum profit, the elimination of accidents as far as the conditions permit, and the stabilizing and coordination of separate operations are the more important objectives, the accomplishment of which determines the efficiency of the management.

Labor Output.—Labor outputs vary between wide limits not only with respect to different kinds of mining but for different mines of the same type. They will differ in a single mine not only with respect to the position of the working in the mine but also in respect to time. Table 123 gives a number of examples of average tons per shift output. References in the bibliography will give additional information. Table 124 gives distribution of labor above ground and underground and average output per man per day for some of the principal coal-mining states.

Labor Report Forms.—Report forms for the different divisions of the work, which must be filled out either by the individual worker or by the boss in charge of the gang on the completion of the shift, are in general use and serve to give a detailed record of performance. From them the labor standards can be figured. They make the work of the foreman and shift bosses more definite and also serve to keep the superintendent and manager in close touch with operations. Practice varies as to the extent to which such reports are used. It is obvious that an individual report from each worker would entail a large amount of clerical labor in systematizing and summarizing, and in order to reduce this to a minimum the number of reports is made as small as possible consistent with securing the necessary information. Several examples of report forms in use are given in Fig. 302. A summary report for the underground labor distribution per shift is also given in Fig. 303.

TABLE 123[a]

Mine	Method	Average tons per shift underground	Surface	Surface and underground	Labor, per cent total cost
Lead-zinc-silver:					
Park-Utah	Square-set, fill	2.26	17.85	....	66.1
Tintic Standard	Square-set, fill	1.107	7.01		
American Zinc, Tenn	Open stope	10.83	......	....	71.1
Hecla, Idaho	Overhand	3.9	......	....	62.0
Federal, Idaho	Overhand	2.9	......	....	66.0
No. 1 Picher	Open stope	5.91	......	....	63.4
No. 2 Picher	Open stope	8.93	......	....	58.0
Gold:					
Tex-Hughes, Ont	Shrinkage	3.21	8.80	....	59.5
Argonaut, Calif	Square-set, fill	1.92	......	....	72.2
Elkora, Nev	Shrinkage	3.58	......	....	61.8
Central Eureka, Calif	Square-set, fill	1.76	......	....	69.3
Copper:					
Champion, Mich	Incline cut, fill	2.64	......	....	66.0
Conglomerate, Mich	Open stope, stull	3.06	......	....	69.8
Tennessee Copper Co	Sublevel, shrinkage	6.08	......	....	65.6
Iron:					
No. 4, Marquette	Shrinkage	6.75	19.83	5.03	59.8
Hard hematite, Marquette	Open stope	6.10	......	....	65.6
Soft hematite, Menominee	Sublevel	10.19	......	....	62.0
Eureka-Asteroid, Gogebic	Sublevel caving	5.71	......	4.43	66.0
Mineville, N. Y	Open stope	7.46	......	....	60.4
Mesabi, Minn	Top slice	7.49	180.95	....	61.5
No. 5, Marquette	Top slice	8.87	36.43	7.13	58.2
No. Marquette	Top slice, sublevel	7.77	......	....	60.8

[a] *Information Circulars U.S. Bur. Mines*, where not mentioned proportion of surface labor chargeable to direct mining is evidently included.

Miscellaneous Reports.—Report forms are used for the different shops. Usually these segregate the kind of work and the number of men on each task. Figure 304 is a report form used for workers employed in the operation and repair of the mechanical equipment.

TABLE 124.—Bituminous Coal Production in 1938 (*Minerals Yearbook*)

Price area number	Number active mines	Total production, tons	Number of employees				Man days labor (million)	Average tons per man-day
			Underground	Surface		Total		
				In pits	All others			
1	3,082	238,454,309	270,932	1,593	41,217	313,742	55.51	4.63
2	1,275	67,141,502	49,574	4,015	12,067	65,656	9.51	7.06
3	260	11,924,952	19,744	91	2,908	22,743	4.06	2.94
4	104	1,538,577	4,042	86	705	4,833	5.50	2.80
5	361	7,872,146	7,112	1,631	1,649	10,392	1.67	4.72
6	253	6,908,022	8,881	10	1,887	10,778	1.78	3.87
7	124	8,152,066	5,815	44	1,645	7,504	1.28	6.36
8	180	2,098,157	685	367	368	1,420	0.25	8.52
9	79	2,732,050	1,126	40	359	1,525	0.27	10.28
10	60	1,722,983	2,093	.....	647	2,740	0.45	3.80
Total	5,777	348,544,764	370,004	7,877	63,452	441,333	71.33	4.89

NOTE.—Minimum price areas as defined by Bituminous Coal Act of 1937 include principal producing districts as follows: 1. Pennsylvania, Northern West Virginia, Ohio, Michigan; 2. Western Kentucky, Illinois, Indiana, Iowa; 3. Southeastern; 4. Arkansas and Oklahoma; 5. Southwestern; 6. Colorado and New Mexico; 7. Wyoming and Utah; 8. North & South Dakota; 9. Montana; 10. Washington, Oregon, Alaska.

Surface Plant.—Upon the general design and arrangement of the surface plant depend the convenience and efficiency of the service rendered. The surface plant provides for power generation, tool sharpening, blacksmith work, repair work, timber framing, storage of supplies, fuel, and ore, ore and waste handling, change quarters for the workers, and office accommodations for the staff and foremen. In addition, ore sorting, crushing, picking, and treatment may be required. The plant usually consists of a number of separate buildings, each of which houses one or more units.

Power is used for hoisting, pumping, drilling, ventilation, the operation of the machinery in the shops, illumination, and the operation of the ore-treatment machinery. In some instances steam power is used throughout the surface plant; but for most plants it is more economical to use steam power for hoisting and the generation of electric power, which is distributed throughout the plant and underground by means of power cables. Electric motors are used wherever power is required except for hoisting service. For drilling, the compressed air is furnished by a steam-

MACHINEMAN'S REPORT

Level.............Place.............
Labor, hours--
Machineman.............Helpers.............
Kind and Size of Machine.............
Number of Holes Drilled.......Feet Drilled.............
Number of Drills--
Received.......Dulled.......Broken.............
Dynamite Used.......Sticks, Fuse.............ft.
Caps.............Number
Actual Drilling Time.............
Time Barring Down.............
Time Lost on Account of--
Broken Machine.......Waiting for Steel.............
Missed Holes.............

Kind of Work

Stoping, Blockholing, Raising in Ore--Waste
Drifting in Ore--Waste, Shaft Sinking

Special.............
(Describe Special Work)
Date.............Shift.............
Name.............
Machine Man
O.K.............
Shift Boss

SHOVELER'S REPORT

Level.............

Name or No. of Stope or Drift	Ore Name of Cars	Name of Shovelers
Waste		

Date.............Shift.............
O.K.............
Shift Boss or Tabman

TIMBERMAN'S REPORT

Level.............*Place*.............
Labor, hours--
Timberman.............*Helpers*.............
Miners.............*Laborers*.............
Number of Sill Floor Sets Placed.............
Number of Second Floor Sets Placed.............
Number of Tunnel Sets Placed.............
Number of Relief Posts Placed.............
Number of Diagonal Braces Placed.............
Lagging Used--
6 *ft*.......12 *ft*.......*Sawed*.............
Number of Slabs Used.............
Number of Wedges Used.............
Number of Chutes Placed--
2½ *inch*.............4 *inch*.............
Other Timber Used.............*lin. ft.*
Spikes Used.............*Drift Pins*.............
Remarks.............
....
Date.............*Shift*.............
Name.............
Timberman
O.K..............
Shift Boss

TOOL PACKER'S REPORT

Level.............
Drill Steel *Received* *Returned*
Hollow-large.............
Hollow-small.............
Solid-large.............
Solid-stoper.............
Picks.............
Rock Hammers.............
Striking Hammers.............
Axes or Adzes.............
Saws, 3½ Ft., one-man.............
Saws, Hand.............
Other Tools.............
....
Time engaged in Tool Carrying.............*hrs.*
Time engaged in other work.............*hrs.*
Remarks.............
Date.............*Shift*.............
Name.............
Tool Packer
O.K..............
Shift Boss

FIG. 302.—Miner's report forms.

driven compressor and distributed underground by a system of pipes. Small air-compressor units are compound steam and double-stage air. Usually compressors of the straight-line type are used. For maximum economy they should be operated with condensers. Large compressors are triple-expansion steam and three-stage air and are always used with condensers. Small electric-driven air compressors are two-stage compressors driven by a silent-chain drive, the motor and compressor being mounted on a common bedplate. Large compressors are of the two-stage type and are driven by belt or are direct-connected. Compressor capacity is figured from the number of drills in normal operation at one time, due allowance being made for altitude. The ratings of compressors and drills and altitude factors are given in manufacturers' catalogues. Air receivers are placed on the surface outside of the compressor house.

SHAFT NO. ______________ **SHIFT REPORT**

	WORK DONE THIS SHIFT			RECAPITULATION OF WORK DONE IN STOPES AND DEVELOPMENT											
	IN STOPES	DEVELOPMENT	GENERAL												
OCCUPATION	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts
Shift Boss															
Machine Men															
Machine Helpers															
Hand Miners															
Muckers															
Stope Sweepers															
Trammers															
Timbermen															
Timber Helpers															
Nippers															
Pipe and Track Men															
Cagers															
Topmen															
Car Greasers															
Powdermen															
Scavangers															
Mud Ball Men															
Timber Rustlers															
Shaftmen															
TOTAL															

Remarks

Date
Name

FIG. 303.—Shift report form.

Where power is furnished from a central plant or by a hydroelectric plant the hoist, compressor, and entire mechanical plant are driven by electric motors. The ease of transmitting and distributing electric power makes it especially suitable for mine service. Steam is sometimes used for pumping, but modern installations favor the use of electric power. Power requirements for different kinds of mining are given in Table 125.

The selection of power agent, whether water, coal, wood, fuel oil, gasoline, natural gas, or producer gas, depends upon availability and costs.

DISTRIBUTION OF LABOR IN MACHINERY DEPARTMENT.

	No. 1 SHAFT	No. 2 SHAFT	No. 3 SHAFT	TOTAL
MACHINERY REPAIRS AND CONSTRUCTION. No.				
On Air Drill Repairs				
Machinist on Repairs				
Machinist on New Work				
Machinists' Helpers				
Boilermakers				
Boilermakers' Helpers				
Pipe and Chain Gang				
Electrician				
Apprentice				
MEN ON ENGINES AND COMPRESSORS. No.				
Hoisting Engine, Top				
Hoisting Engine, Underground				
Hoisting Engine, Auxiliary				
Engine Wiper				
Compressor				
Waste Haulage				
Ore Loading Engine				
MEN ON BOILERS. No.				
Firemen				
Boiler Cleaner				
PUMP MEN. No.				
Mine Pumps, Station				
Mine Pumps, Sinking				
Washer Pumps				
Head Pumpman				
BLACKSMITH SHOP. No.				
Blacksmiths				
Blacksmiths' Helpers				
SPECIAL HELP.				
Haulage System				
Cables				
Cleaning, Sorting, etc.				
Total Men				

FIG. 304.—Report form for surface and underground labor employed on machines and repairs.

If fuel and suitable boiler water are available, then steam- and turbine-driven electric generators are employed and electric power is distributed where needed. Many large mining plants are thus equipped. If fuel is costly and boiler water unsuitable, diesel engines employing fuel oil are installed, electric power being generated and distributed. Mines in the Southwest and in more or less isolated districts have adopted diesel-engine units as the principal source of power. In some cases where there is a local wood supply, producer-gas generators, gas engines, and electric generators are used. In small mines, where power is required during prospecting and exploration, gasoline engines and small diesels with electric generators have been found to be convenient and as economical as the conditions permit. In a number of such cases the portable type of gasoline-driven compressor and gasoline hoist serve for shaft explora-

TABLE 125[a]

Mine	Kilowatt-hours per ton mined					
	Air compression	Hoisting	Pumping	Ventilation	Lighting	Total
Lead-zinc-silver:						
Park-Utah	6.68	6.94	12.63	0.99	1.99	29.23
Tintic Standard, Utah	14.90	15.40	......	11.60	4.70	46.60
Cortez, Nev	21.18	9.37	0.93[b]	.....	0.29	31.76
Mascot, Tenn	3.81	1.06	4.04	0.05	0.63	9.68
Federal M. & S., Idaho	9.42	2.73	2.25	0.43	0.24	15.07
Pecos, A. M. Co., N. Mex	6.88	1.30	2.73	.....	0.87[c]	11.78
No. 1 Picher	6.00	2.77	0.22	.....	0.92[d]	9.91
No. 2 Picher	2.51	2.42	0.82	.....	.....	5.75
Gold:						
Tex-Hughes, Ont	13.55	5.10	0.54	.....	0.37	19.56
Argonaut, Calif	9.02	9.90	6.40	2.11	0.39	33.08
Elkora, Nev	8.48	1.85	0.51	.....	0.09	10.92
Copper:						
Champion, Mich	12.40	4.00	4.70	.....	1.00[e]	21.70
Old Dominion, Ariz	22.80	13.10	34.20	6.60	2.70[d]	79.40
Matahambre, Cuba	8.00	3.00	1.79	1.22	.....	14.01
Iron:						
No. 4, Marquette	8.20	1.82	2.51	.....	0.80[d]	13.83
No. 1, Menominee	5.90	3.64	1.89	.....	0.47	11.80
Eureka-Asteroid, Gogebic Range	5.63	3.91	4.67	.....	1.41	15.62
Mineville, N. Y	5.16	4.64	0.26	.....	0.85[f]	10.91
No. 5 Marquette	4.49	2.78	3.96	.....	0.60[d]	12.49
No. 2 Marquette	4.24	2.30	1.78	.....	0.94[d]	9.39

[a] *Information Circulars, U.S. Bur. Mines.*

[b] Motor haulage.

[c] Timber framing.

[d] Includes motor haulage.

[e] Ventilation and lighting.

[f] Haulage and tramming. Miscellaneous uses included in total.

tion satisfactorily. For adit exploration the portable gasoline-engined compressor and a gasoline engine for the operation of a ventilating fan meet the situation. Many Western mining districts are served by transmission lines distributing power from hydroelectric plants, and where this is the case electric motors are used for the operation of air compressors, hoists, pumps, ventilating fans, and shop machinery. Such power costs are usually lower than those for which power can be generated in small plants. Large copper mines, where smelting is a feature of ore treatment, make use of waste-heat boilers and turbine generators for part or all of their power as in the case of the Magma and United Verde mines in Arizona. Coal mines either purchase power from central power plants or generate their own electric power in steam power plants and distribute it to motors placed where convenient. Electrical power is now used for almost all mine purposes both on the surface and underground in large enterprises. Rock drills and some mechanical loaders alone require supplementary power in the form of compressed air. Air compressors are driven by electric motors.

Tool sharpening in a hard-rock mine is an important feature, and a suitable equipment of power tools is essential. Hand drills, machine steel, bars, picks, and moils are the usual run of tools requiring sharpening. They are distributed to the working places by the miners or tool packers and are gathered and sent to the shop at the end of the shift. Oil forges, power sharpeners, power hammers, and cooling tubs are the usual features of the equipment. Specialized tools for sharpening, hardening, and re-forming of detachable bits are included.

Blacksmith work for repair and new work is required to a greater or less extent about almost all mines. Tool sharpeners when not occupied upon the sharpening work turn their attention to the repair work. Cages, skips, cars, track, switches, and chutes are the principal features requiring the attention of the blacksmith. The tool equipment consists of forges, anvils, power hammers, punches, plate cutters, and small tools.

Repairs requiring machine tools and machinists are necessary upon machine drills, pumps, cars, electric locomotives, motors, ventilating fans, steam engines, etc. The tool equipment of the machine shop comprises a heavy engine lathe, a light lathe, a radial drill press, pipe threader, bolt threader, power hack saw, emery wheel, bench tools, and sometimes a slotter or milling machine. Provision is made for the repair of electrical machinery. The size of the mine and the number of mechanical appliances determine the importance of and the extent to which the machine shop is equipped. The principal advantage of a well-equipped machine shop is in the saving of time in making repairs. It is often more economical to make repairs at a mine than to send to large centers for repair parts. For standard machines it is, however, usually more economical to purchase repair parts. At very large mines a foundry is often maintained as an adjunct to the machine shop.

The timber-framing shop is of importance where timber is used for support. The usual equipment is a swing saw for end cutting, a power saw for wedges and small work and, where the square-set system is used, a timber-framing machine. In some mines sawmill equipment is included and logs are squared and cut into boards and timbers. Where timber is treated with preservatives a treatment plant is added.

Storage for supplies, timber, fuel, and ore is required. Timber is stored in open yards, miscellaneous supplies in storehouses, powder and accessory explosive material in separate magazines, heavy supplies in racks, and ore in bins or in yards near the shaft.

Change quarters for the workers are provided by most progressive mines. The usual equipment consists of individual lockers, shower baths, and washbasins.

Office facilities for foremen, shift bosses, and timekeeper are usually placed close to the shaft, while a separate building is provided for the engineers. The foreman's office is equipped with a counter, desk, lockers and shelves, change room, and shower. The engineers' office building is divided; a separate room equipped with counters, desks, files, and lockers is placed at the disposal of the accountant and his clerical force, and another, equipped with drafting tables, desks, lockers, etc., is placed at the disposal of the surveyor and mining engineer. In addition a change room provided with lockers and showers, a blueprint room equipped with printing frames and a darkroom, and a fireproof vault for the safekeeping of books, records, and maps are often added. Where the manager maintains his office at the mine, additional rooms are provided for in the staff office building. The assaying laboratory is placed in a separate building.

Ore-picking and crushing equipment are placed in close proximity to the mine outlet. The ore-treatment plant may be close to the mine outlet or at a distance, depending upon the proximity of a suitable site. Careful consideration in the selection of mine and ore-treatment plant sites is given to access to the mine shaft and to the main line of communication, road or railroad, as well as to the presence of dumping space of sufficient area and slope for waste and tailing dumps. The individual buildings are constructed of wood, timber frame and corrugated galvanized iron covering, steel frame and corrugated iron covering, steel frame and concrete, or reinforced concrete. The permanency of the plant and the capital available determine the type of building. Where the life of the mine justifies it, fireproof construction is highly desirable, and in all cases a minimum of inflammable material should be used. The shaft should never be covered by a building except one of fireproof construction. The timber frame with galvanized-iron covering is largely used and satisfies the requirement of a minimum of combustible material.

The various buildings should be grouped about the mine opening compactly and in such a manner as to admit of convenient access to the mine outlet. Figure 305 represents a flow sheet, which shows the essential features in the movement of supplies, tools, repairs, ore, and waste. The preparation of such a flow sheet is a necessary preliminary to the design of the surface plant in a particular case. The individual sites of the buildings will be more or less controlled by the topography in the vicinity of the mine outlet. The space allowances between the buildings are determined by the fire limits established and range from 30 to 50 ft. Figure 306 shows a design suitable for a flat site. Where the building construction is not fireproof, separate buildings as shown in the figure and ample fire protection are advisable, but where fireproof construction

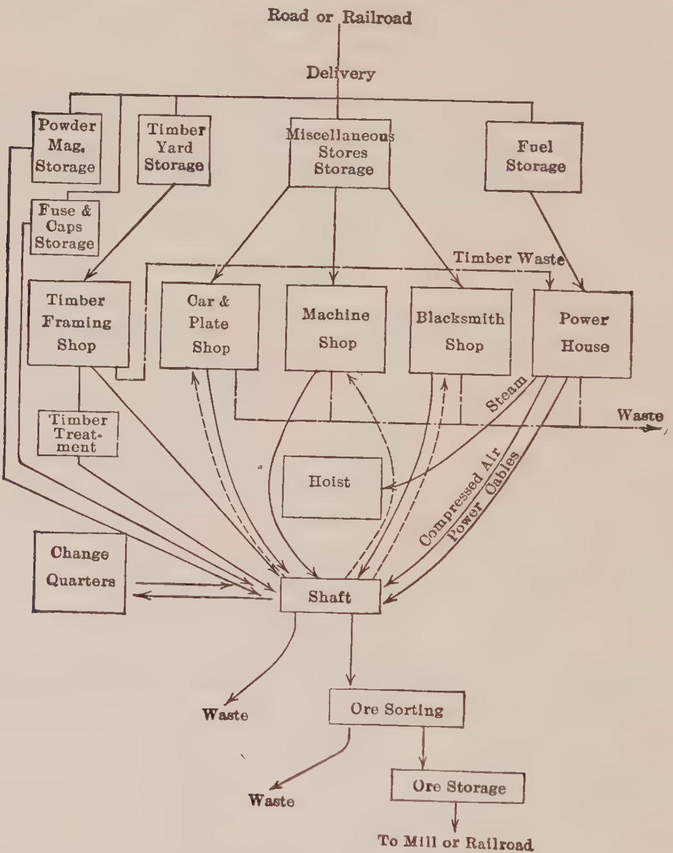

FIG. 305.—"Flow sheet" of mine surface plant.

is used the units may be concentrated in fewer buildings and the cost of construction thereby reduced.

Each unit of the plant should be separately studied by analyzing the routine steps in the handling of the materials. The mechanical appliances should then be grouped so as to provide for the movement of the material through the sequential operations required. The delivery of the supplies required and the final delivery of the finished product to its destination should be provided for. The lines of the building can then be determined. Lighting, water supply, sewerage, heating, and

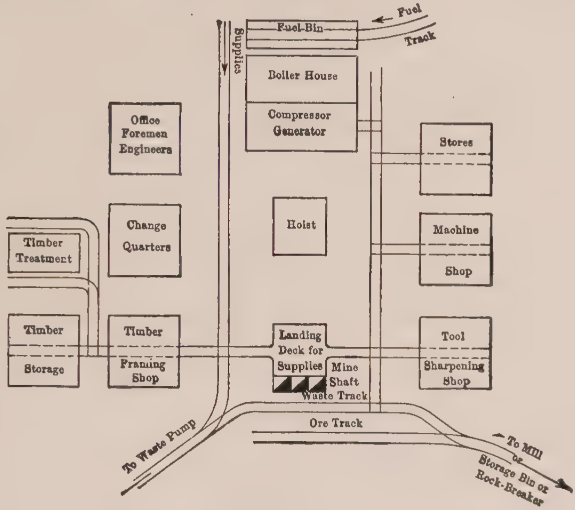

FIG. 306.—Plan of mine surface plant.

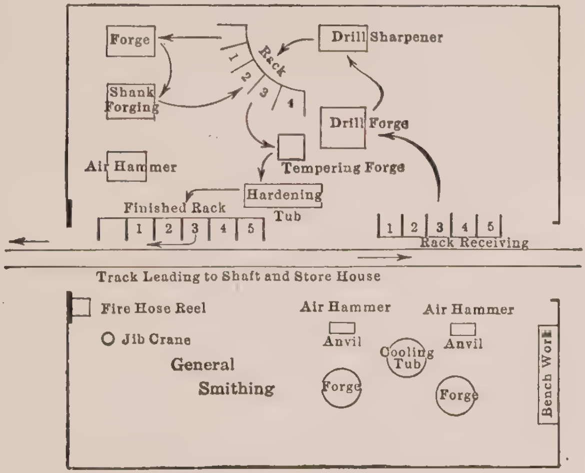

FIG. 307.—Plan of mine blacksmith shops.

fire protection should be separately considered for each building. In Fig. 307 the plan of a tool-sharpening and blacksmith shop is illustrated. The routine of sharpening, in sequence, is as follows: Delivery of drills from shaft, sorting into different lengths, heating to a forging temperature, forging by machine sharpener, placing in rack, heating for hardening, quenching in water, placing in finished rack, and delivery to mine in assorted sizes. Supplementary steps not required on all drills are shank forming and the welding of new steel to old shank ends. The arrows in the illustration indicate the sequence of the routine and supplementary steps. Where the use of detachable bits predominates, both equipment and routine are different from that which prevails where solid steel is in use.

Underground Plant.—Many details of the underground equipment have been given in preceding chapters and require no repetition. All underground work is analyzed and standardized as far as practicable, and appliances and equipment are selected in accordance with the standards established. Certain features of the underground plant have not been discussed and will here be considered.

The compressed-air pipe system consists of a main supply pipe in the shaft. A 4-, 6-, or 8-in. pipe, depending on the number of drills in operation, is placed in the pipe compartment of the shaft. At each level a branch connection and valve are placed. This may be a 2-, 2.5-, or 3-in. pipe. At the station a small valve is tapped into the branch air pipe for emergency use. In some mines it is the practice to install an air receiver at each station. The branch pipe is extended along the level and a tee is placed at each raise. Unions are also placed at regular intervals so as to permit of disconnecting the pipe. At the raises 1- or 1.5-in. pipes are connected. On exceptionally long pipe runs receivers are installed at an intermediate point.

In some mines electrically driven air compressors are installed underground with the object of avoiding long pipe runs. Under such conditions air losses are reduced to a minimum and greater economy results.

Electrical power cables, lead-incased and armored, are used in the shaft. Alternating current of a maximum voltage of from 2,000 to 2,200 is frequently transmitted underground and presents no practical difficulties. Transformers are placed at the station or close to the motors. They should preferably be placed in a fireproof chamber. The voltage used in the distribution may be 220, 440, or 500, and well-insulated distributing leads are essential. Where direct current is required it may be generated at the surface and conducted underground by heavy insulated leads, or an underground motor-generator set, using alternating current, may be used as a source of supply. The voltage supplied is usually 250.

At one time large mines established drill-sharpening stations underground so as to minimize the time and labor required to handle the relatively large amount of drill steel to and from the working faces. The detachable drill bit has made such stations unnecessary. Underground drill-sharpening shops have disappeared, and all sharpening work is now done in surface shops. Drill repairs are likewise made in a drill-repair shop on the surface.

Water is supplied for drinking and for drilling by an underground pipe system or by distributing it in barrels to the different levels. A proper sanitary arrangement should be installed for the drinking-water supply.

Water closets are installed at each level, and for this purpose closed cars or abandoned workings are utilized. Such devices and places should be regularly inspected and kept in a clean condition.

Telephones should be installed at the more important working places on each level as well as at the shaft stations. Such equipment applies more particularly to large mines.

Fire protection in timbered mines is an important feature. Either a water system is installed and hose reels are placed at critical points, or hand fire extinguishers are placed at stations and in heavily timbered stopes. Fire inspection is a necessary adjunct in mines of this kind.

Each level should be marked with suitable signs indicating the direction to safety outlets and ladderways. Fire doors should be placed at suitable points on each level to aid in the control of a possible fire. At each station there should be placed a bulletin board for recording misfires or other dangerous conditions in the workings. A first-aid kit should be placed at each station. Ladderways and exits should be regularly inspected and kept in good repair. Refuge chambers, each provided with a steel door and a gas- and fireproof stopping, with relief valves at the top of the chamber operated from within the chamber, and an independent compressed-air supply as well as a water supply, are an additional safeguard underground.

Control of Mining.—An accurate map of all the mine workings is maintained. The mine surveyor has this specific duty. He measures all new development work, shafts, drifts, crosscuts, raises, and winzes once each week and makes the necessary additions to the mine maps. Once each month the stopes are cross-sectioned and the advance is determined. Charges in the formations are noted. All working places, chutes, etc., are numbered so that accurate records of the work done in the mine can be made. The number of cars of ore drawn from each chute is recorded on a tally board and reported each shift. The cars of waste delivered to each stope are also recorded. An assay map showing the location of the samples is kept up. Samples of stope faces, drifts, etc., are taken at frequent intervals and the results entered on the assay map. Stoping limits are established either by the walls of the vein or, where these are indefinite, by samples taken from drill holes. As ore bodies are developed tonnages and average values are computed. An ore account is maintained, of which the items at the end of any given time period are:

Ore reserves in tons at beginning of period.
Ore reserves in tons added by new development during period.
Ore mined during period.
Net ore reserves at the end of the period.

In a coal mine and in many iron mines the mine may start upon its career with a known tonnage, and the possibilities of increasing the tonnage may be remote. The mining problem is one of keeping the costs down and the grade of the ore up. In a precious-metal mine the quantity and value factors are uncertain and as a consequence additional tonnage must be constantly sought. Frequent samples must be taken in order to prevent unprofitable material from going to the treatment plant. The ore sent to the treatment plant must be maintained as nearly as possible at a uniform grade. This is done by varying the quantity of ore taken from the different stopes. The prospecting problem is just as important as any feature of the mining operations. Intensive studies of the geology and nature of the deposits are made with the object of getting suggestions for new work. Mine models are made, and these are consistently studied in order that a better understanding of the deposit can be obtained. New

Working, number or description	No.	No.	No.	No.	No.
Cross section					
Length					
Rate, feet per shift					
Number of shifts to complete					
Date started					
Date finished					
Cost per foot, estimated					
Total cost					
Cars of ore					
Cars of waste					
Average tramming distance					
Estimated value of ore					

ground is tested out by diamond drilling or crosscutting. The manager, mine superintendent, geologist, surveyor, and foreman confer on the possibilities of working in certain directions.

Footage and tonnage estimates are prepared in advance of operation and serve the purpose of synchronizing development and stoping. Such estimates are made either upon charts or in the form of tables. A footage-estimate table is shown on page 650.

The features in the control of open-pit mining as exemplified by the practice upon the Mesabi Range are as follows: The construction of cross sections of the ore body from informations obtained from drill holes and underground workings; the construction of an accurate topographical map established upon a coordinate system, the unit of which is a 20-ft. square; monthly cross sections of the stripping pit at 40-ft. intervals and the computation of the yardage excavated from the cross sections; cross-sectioning of the stripping pit at the end of each 6 months' interval and the determination of the final estimate of yardage for the period; an annual cross sectioning of the ore pit at 40-ft. intervals and the computation of the ore excavated. The railroad car weights give the ore production for any time period. The grade and class of ore are determined from its position upon the cross sections of the deposit that show the variation in the per cent of iron and phosphorus. The grade of ore shipped is checked by car samples.

A tonnage estimate for stoping is given in the table on page 652.

The control work in an underground Mesabi mine involves the following procedure.

1. All workings, development, and ore mining are measured up twice a month and the underground maps brought up to date.
2. The tonnage supplied by contractors is estimated by the number of cars, and payment is made at a given rate per mine car. Where a single contract loads into a chute, the trammer boss reports the number of main haulage cars loaded from the chute. Where several contracts use a chute in common a chip boy is stationed at the chute and receives from each contract a brass check for each car dumped. The brass check is numbered with the contract number. The chip boy reports the number of cars from each contract at the end of the shift. As a check the trammer boss reports the number of cars drawn from the chute on the main haulage level.
3. The mine tonnage is checked by determining the average weight of a skip load (railroad weight tonnage divided by the number of skips hoisted) and the number of skips hoisted. The average skip load is reduced about 15 per cent for winter hoisting on account of the freezing of a layer of ore on the sides and bottom of the skip.
4. The grade and class of ore mined are determined from the position of the working and the cross sections of the ore body that show the grade and class of ore throughout the deposit. Drifts are sampled every 25 ft. and raises every 5 ft. and the results of the analyses recorded on the cross sections of the ore body.

In coal mining the mining control consists in the making of semi-annual surveys of the workings and monthly surveys in which room and gangway centers are established. These surveys are concerned with the first mining or the driving of entries and rooms. The second mining requires the maintenance of a pillar map, on which each rib or pillar is numbered and the monthly progress of robbing is shown. Where filling is used the progress of filling is checked up each month and shown upon the map, for which purpose a separate map may be required. The yardage driven by the miners on contract is measured once each month. Where portions of the coal area are worked on royalty the area and tonnage are determined on the monthly or semiannual surveys. On small areas the foreman makes the returns, and these are checked by the engineers. Tonnage estimates of room, entry, and pillar coal, net coal remaining, and coal produced for a given time period are made. The estimate is made by planimeter measurement upon the map and measurements of the thickness of the coal seam. The output of the individual miners or groups of contractors is determined by weighing the cars as they are delivered to the tipple. Each car is numbered by means of a brass check,

Stope or block number	No.	No.	No.	No.	No.
Estimated tonnage					
Estimated average value					
Total value					
Number of units breaking					
Number of shifts required to mine					
Cost of breaking					
Number of chutes required					
Cost of chutes					
Estimated waste for filling					
Estimated timber and waste cost					
Cars per shift output					
Average tramming distance					
Tons output per shift					
Value output per shift					
Estimated cost per ton					
Date started					
Date finished					
Mining method					

SHAFT NO. LEVEL PLACE MONTH OF 191

	Machine Men	Machine Helpers	MACHINE MEN'S REPORTS																Hand Miners	Sorters	Muckers	Stope Sweepers	Trammer	TRAMMERS' REPORTS				Pipe and Trackmen	Timber Men	Timber Helpers	
			HOLES DRILLED			STEEL		POWDER		FUSE		BREAKAGES			Time Mucking	Time Barring Down	Time Timbering	Missed Slots						CARS HOISTED			Waste to Stopes				
			No.	Depth	Hours	Rec'd	Drilled	Rec'd	Used	Rec'd	Used	Machines	Drills	Timber										Ore	Mill Dust	Waste					
1																															1
2																															2
3																															3
4																															4
5																															5
6																															6
7																															7
8																															8
9																															9
10																															
21																															20
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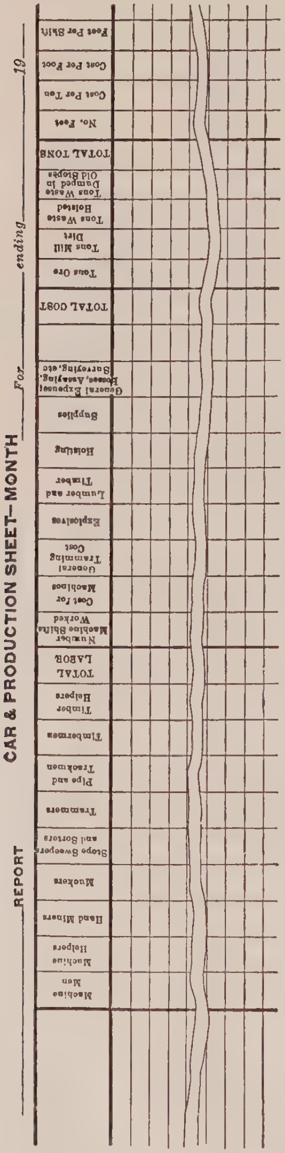
27																															26
28																															27
29																															28
30																															29
31																															30
Totals																															31
Amount																															

LABOR	SHIFTS	COST		COST			PRODUCTION		REMARKS
Machine Men				Total Labor			Tons Produced		Average $______ per Ton
Machine Helpers				Machine Drill Cost $________ per shift			Cost Per Ton		% Ore
Hand Miners				Trimming ________c per ton			Cost Per Ton, Ore Only		Average Large Samples
Sorters				Explosives			No. Cubic Feet Stoped		
Muckers				Timber			Cost Per Cubic Foot		Average Mill Ore Samples
Stope Sweepers				Hositing ________c per ton			No. Linear Feet Driven		
Trammers				Supplies			Cost Per Linear Foot		
Pipe and Trackmen				General Expense ________c per ton			No. Shifts Worked		
Timber Men							No. Feet Per Shift		
Timber Helpers							Explosives, Cost Per Ton		
							Breaking, Cost Per Ton		
TOTAL				TOTAL			Timbering, Cost Per Ton		

FIG. 308.—Monthly production and cost summary for a single working place.

on which the number of the miner or contract is marked. Credit is given each number for the weight of coal in the car. The miners frequently employ a checkweigher to secure accuracy.

Monthly Mining Estimates and Cost Sheets.—Summaries of the development footages, ore production, waste handling, and costs are made each month. There is a wide variation in practice in the details of the monthly summaries. An example of a detailed summary of a given working place is given in Fig. 308. Figure 309 is the monthly cost sheet in use at the same mine from which Fig. 308 was taken.



REPORT ____ CAR & PRODUCTION SHEET—MONTH ____ For ____ ending ____ 19__

Machine Men	Machine Helpers	Hand Miners	Muckers	Stope Sweepers and Sorters	Trammers	Pipe and Trackmen	Timbermen	Timber Helpers	TOTAL LABOR	Number Machine Shifts Worked	Cost for Machines	General Tramming Cost	Explosives	Lumber and Timber	Hoisting	Supplies	General Expenses Bosses, Assaying, Surveying, etc.		TOTAL COST	Tons Ore	Tons Mill Dirt	Tons Waste Hoisted	Tons Waste Dumped in Old Stopes	TOTAL TONS	No. Feet	Cost Per Ton	Cost Per Foot	Feet Per Shift

FIG. 309.—Monthly production and cost sheet.

Control of Ore Treatment.—Picking and sorting plants are usually placed close to the outlet of the mine, and the supervision of the work is placed in the hands of a foreman or boss. The work is controlled by estimating the tonnage either by the number of cars or skips handled or by weighing the product delivered. The weight of the reject is determined by the number of cars of waste sent to the dump or by weighing. Samples are taken at regular intervals and the work of the ore sorters is checked. The summary reports give the weight of the first-class ore, weight of second-class or mill ore, and the weight and value of the waste rejected.

Where a more elaborate method of ore treatment is used, the supervision of the control is in the hands of the metallurgist. The task is a specific one—so many tons of ore per 24 hr. to be passed through certain steps and as high a percentage of value to be extracted at as low a cost as possible. The ore is sampled as it is delivered either by being passed through a mechanical sampler or by some method of hand sampling. The plant is charged with the weight of ore and the total value. It is credited with the value lost in the tailing or waste and the values recovered. The account should balance. Some form of mechanical sampler is used on the tailing discharge. The samples for each shift and each day are assayed or analyzed, or both, and a report is made to the manager. The bullion or concentrates are weighed and sampled, and the values are reported.

The detailed working of the plant, as for example the saving effected by a given machine or group of machines and their cost of operation, is determined by sampling the heads and tails of the machine or group and by measuring the rate of feed. The power required for operation, the supplies, the cost of repairs, and the total costs for the group are also determined. The causes for low extraction are determined by physical or chemical examinations. Thus the efficiency of each individual part of the plant can be determined. Separate cost accounts for the plant are kept, as well as a system of individual reports, which indicate tonnages and values recovered, weights and values of products, and weight and value of the tailing.

Control of Costs.—The direct cost of mining is made up of the cost of labor, supplies, and materials. The unit cost is given on the basis of the ore produced. Unit costs are computed for each month. Thus the manager can tell the direct outgo upon each ton of ore. Direct unit costs fluctuate more or less from period to period but tend to assume a constant figure. Indirect costs include depreciation on plant and equipment, taxes, insurance, superintendence, general office expense, etc. The sum total may or may not be of constant magnitude for a given time period, but the tendency is to approximate this condition. With a variable output the indirect unit costs fluctuate between wide limits, reaching a minimum for a maximum production and a maximum for a minimum production. The amount of prospecting and development will change the unit costs for any given period if these accounts are included in the direct mining costs. Both are best kept separate and should be figured independently in the cost reports. The final measure of efficiency is the cost; hence the details of cost are matters of considerable concern to the manager. Especially is this so if the margin of profit is small. How to reduce costs is a problem to the manager. He can attack the problem in two ways, by endeavoring to secure greater all-round efficiency and by reducing wastes. An accurate system of accounts and close supervision of the enterprise materially assist to this end.

Present-day cost control is complicated by political concepts, such as old-age pensions, unemployment insurance, the stimulation and appeasement of labor organizations, and the general growth of experimental extensions of governmental activities. All must eventually be paid for out of industrial production. The resulting tax load will be overwhelming. Wages and taxes are basic price elements. Wage and tax increases require equivalent price increases. Price fixing aggravates the difficulties of industrial economics.

Pay rolls are subject to various surcharges, such as industrial-accident compensation, a 1 per cent old-age tax, and a 3 per cent unemployment-insurance tax. The accounting expense of an enterprise is greatly increased. The employer pays income taxes, federal and state, property tax, sales tax, school tax, and in some states other special taxes. An example of taxes paid by the mining industry is afforded by the gold mines of the Transvaal. According to *Mineral Trade Notes*, the Government Mining Engineer of the Union of South Africa stated that the revenue derived from income tax and the special contribution from the gold mines totaled £16,759,163 for the fiscal year ended March 31, 1945. During the same period dividends, including the annual redemption allowance of capital expenditures (about £6,500,000), totaled £13,701,000. Excluding the allowance for amortization, the combined dividends, the income, and special levy taxes total approximately £23,960,164, of which the government received nearly 70 per cent.

Experimental Work.—In a large organization the opportunities for experimental work are seriously restricted by the routine work, and it is impracticable in many instances for staff men to carry out any systematic testing of appliances and methods. A separate division to handle such work is established by progressive managements and often results in the attainment of greater economy in the use of supplies, the adoption of more efficient appliances, increased extraction in millwork, the production of valuable by-products, and the simplification of methods.

Miscellaneous.—Contracts for the execution of certain work, for the delivery of supplies, leases, miners' contracts, and machinery contracts form the usual run of business agreements that must be drawn up, authorized, and executed. The contract involves the designation of the two parties, the subject of the agreement, specifications, the money or other consideration, the terms and conditions of payment, the penalties for nonfulfillment, the method for arbitrating disputes, the time limits, date, and signatures of both parties. Agreements are signed by the manager or by any other person authorized by him to act. Originals of all agreements are filed, and the manager designates the staff member responsible for the supervision of the execution of the contract. Contracts of more than nominal importance are drafted by the legal department of the company.

The purchase of supplies is an important detail. A special purchasing agent is engaged to take charge of the division where the amount and money involved are sufficient to justify his salary. Specifications covering quantity, material, quality are drawn up for each supply. Bids are secured on the basis of these specifications, each bid specifying price and time of delivery. Supplies when received are inspected and accepted or rejected in accordance with the specifications. Price is usually influenced by the quantity ordered, so that orders should be so adjusted as to secure the advantage of the lowest prices offered. Some little skill and attention are required to keep the stock of supplies down to the fixed maximum. Usually limit amounts are established for each supply, and these limit amounts are such as to allow a reasonable time to elapse before placing the order. In this way orders can be grouped or concentrated, resulting in economy in placing the order and in shipping and handling the supplies. Heavy supplies used in large amounts are purchased by the carload at regular intervals.

Probably one of the more troublesome details in mine operation arises from the theft of supplies and ore. Theft of supplies, as well as waste, is avoided by a proper system of storage, accounting, and checking. The prevention of ore, amalgam, and bullion thefts requires constant watchfulness and close checking. Only trusted employees should be allowed to handle the valuable end products. Day and night watchmen are necessary at critical places. Unauthorized and casual visitors should be discouraged. Thefts of rich ore in the mine are especially difficult to control. Watchmen are often stationed in the rich stopes. Change quarters are designed so that the workers cannot remove any ore in their clothing or dinner pails without detection. The selling of rich ore and bullion is restricted by legislation in most states and a certain amount of protection is thus afforded mining companies, but there are many channels for the disposal of stolen ore and bullion, and these must be discovered and watched.

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CHAPTER XVII

MINE ACCOUNTING

The essential requirements of a mine accounting system are a record of outgo and income and a record of performance in work or results accomplished, together with the accompanying costs and periodical summations of the financial condition of the company or property. Accuracy, clearness, simplicity, and applicability to the situation should be the characteristics of the system. While there are many features in common in different accounting systems, there are great differences in detail. No uniform system is applicable to all cases. A careful analysis of the demands that must be met by manager, staff, and workers and the limiting conditions of the particular case are necessary before a satisfactory system can be planned. The accountant and manager and his staff of engineers should cooperate in working out the details of the system. The system should be elastic enough to take account of varying conditions as they arise. The initial division of the accounts requires the following subdivisions, and each will be separately considered.

The labor account
The supply account
The cost account
The sales of product account
The cash account
The profit and loss account
The balance sheet

The Labor Account.—Labor timekeeping is kept either by the time-sheet or the time-card method. The simplest application of the former requires each worker to go past the timekeeper's window and verbally give his name or number to the timekeeper, who checks against the time sheet. When the worker reports off shift he again goes past the timekeeper and gives his name or number, which is checked off. The time sheet is signed by the timekeeper and given to the accountant. The simplest application of the latter method requires each worker to receive a time card, on which his name and number may be stamped when he reports for work. When he reports off shift he fills out the time card with the number of the working place and the particular kind of work and hands it to the timekeeper. While both of the above-described methods are applicable in certain instances, a somewhat more elaborate method is more common. The method used by the Crystal Falls Iron Mining Company requires the following steps.

1. Men on going to work report their brass-check numbers and the timekeeper records each number.

2. Men going off shift report their numbers and the timekeeper records them.

3. During the shift the timekeeper makes one round underground and sees each man at work. He records the working place and classifies the worker's time in accordance with the separate accounts. He makes a similar round twice each shift on the surface.

4. The timekeeper's daily report is given in the accompanying form.

Daily Labor and Production Report

Date——————19———

Surface	Total	Underground	Day	Night	Total
(1) Office force		(1) Mining captain and ass'ts			
(2) Mining eng.		(2) Mine foreman			
(3) Master mechanic		(3) Miners			
(4) Machinists, etc.		(4) Miners—contract, etc.			
(33) Total men		(21) Total men			
		(22) Product per man			

(23) Total men employed............
(24) Product per man............
Number of cars trammed............
Number of skips hoisted............
Number of tons hoisted............
Total product to date for month............days............
Average daily product for month............
Estimate for month............days............
Cars shipped from stock pile............
(25) Number of cars trammed { No. 1 shaft............ / No. 2............
(26) Number of skips hoisted { No. 1 shaft............ / No. 2............
(27) Number of skips rock hoisted............
(28) Tons of ore hoisted.........

As a check upon the timekeeper's daily report the surface foreman and underground shift bosses are required to take each man's time as they make their rounds. Each submits a report, which is sent to the superintendent and which checks the monthly totals on the pay roll.[1]

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[1] J. D. Vivian, *Trans. Lake Superior Min. Inst.*, vol. 16, p. 70.

The time sheet used at a coal mine is given in the following:

Daily Time Sheet——————————Mine

Date——————————

No.	Started		Quit		Time, hours	Rate	Amt. earned	Charge to	What doing
	A. M.	P. M.	A. M.	P. M.					

The daily time sheets or time slips are used for the preparation of the pay roll. The pay roll is a summary of the amounts earned each day by each worker or the hours of labor each day. The form in common use is given below.

Payroll————Mine——————Month——————19——

No.	Name	1	2	31 spaces here. Total shifts or hours	Rate	Amount earned	Deductions	Balance
Total								

For contract working where the unit of payment is the linear foot, car, or ton, the entries would correspond, and in place of total shifts would appear footage, tonnage, or number of cars. For bonus working a similar change would be necessary. For employees on salary a separate pay roll is usually made up.

Payment of wages or salaries is customarily made by check. The checks are made out, verified, and distributed by the cashier or his assistants. In some mines it is the practice to require workers to sign the pay roll as a receipt of payment made. The same practice applies to the salary pay roll.

Segregation of labor costs on the basis of underground, surface, transportation, ore-treatment, and construction work follows. These may be termed primary accounts. Each primary account may be and usually is divided into a number of secondary accounts, under each of which the charge for labor is entered. Thus under underground work the secondary accounts may be drifting, crosscutting, raising, stoping, timbering, shaft sinking, tramming, waste filling, etc. Under surface work the secondary accounts may be tool sharpening, blacksmithing, timber framing, hoisting, machine shop, power plant, etc. Secondary accounts may be still further divided and each subdivision charged with its appropriate amount and cost of labor. A definite system of accounts is laid out, and each account is given a number. The timekeeper's daily report is prepared in accordance with the account numbers, and the work of the accountant consists in entering the quantities in each account. The separate accounts are totaled each month and the labor cost is determined.

Pay-roll accounting is now complicated by the incidence of various taxes, such as social security, unemployment insurance, and withholding. Both employer and employee are subject to a 1 per cent pay-roll tax for old-age pensions, and the employer is subject to a 3 per cent tax for unemployment insurance. The employer deducts the withholding tax from the employee's pay-roll account. The employer also pays the industrial-accident-compensation insurance, based on a percentage of the pay roll. He may also have to deduct employees' union dues and pay them to the union organization. It is necessary to maintain accurate accounts of all such transactions and to make quarterly and annual reports to the government. Deductions from employees' accounts are usually entered upon the pay-roll check or upon the envelope containing the cash payment.

The Supply Account.—Three separate subdivisions are usually necessary, a ledger, a storekeeper's stock, and a freight account. In addition, three files are required for orders placed, bills received and checked, and bills paid. The ledger account comprises a separate debit and credit account for each firm from which supplies are purchased. Bills are received, shipments checked, prices and totals checked, and the bills filed as ready to be paid. The amount is entered in the ledger. Each month the separate accounts are settled by check. The supplies are charged to the storekeeper's account after adding the freight and handling charges, which are apportioned as accurately as possible.

The storekeeper's account is a modified ledger account. Supplies are delivered on a requisition, which shows the account to which the supply must be charged. The storekeeper charges the individual account with quantity and cost and credits it with the quantity and cost of any supplies returned. On a separate sheet supplies received are charged against the storehouse and supplies issued credited to it. The balance of the latter account is represented by the stock of supplies carried.

The account of stock is maintained by a system of cards, each card being a record of a single supply. The form used upon the card is given below. Balances are written in red ink.

Article————— Card No.—————

Date	Balance	Received	Cost	Unit cost	Used qt.	Value	Account charged	Notes

Daily, weekly, or monthly reports of the supplies issued and the account to which they are to be charged are prepared by the storekeeper and submitted to the accountant, who makes the necessary entries against each account. Semiannually or annually the entire stock of supplies is inventoried and the balances on the stock cards are checked.

A separate cash account is required where mine stores are sold or where settlements are made each month. A ledger account for each purchaser is kept. The storekeeper reports cash sales and prepares monthly statements of the ledger accounts.

The Cost Accounts.—The cost accounts serve the important purpose of keeping the management and each department head in close touch with the detailed expense involved in the working operations. The extent to which segregation of costs is necessary will vary in each individual mine. Modern office facilities, adding and multiplying machines, typewriters, and time stamps have made possible without excessive

TABLE 126.—ANALYSIS OF SEGREGATION

General	Mining and development	Accessory	Repair	Preparation
Labor	Stoping	Machine drilling	Machine drills	Timber framing
Supplies	Drifting	Breaking	Cars	Timber treatment
Power	Crosscutting	Shoveling	Locomotives	
Repairs	Raising	Chute loading	Ropes	Tool sharpening
Depreciation	Winze sinking	Tramming	Pipes	Storehouse
Supervision	Shaft sinking	Haulage	Track	Change house
Various indirect items of expense	Diamond drilling	Pumping	Ventilators	
		Ventilation	Hoist	
	Ore treatment	Illumination	Electrical equipment	
	Conveying	Hoisting	Signal system	
	Crushing	Timbering	Power distributing system	
	Sizing	Surface transportation		
	Concentration	Stock piling	Power plant	
	Classification	Stock pile loading	Buildings	
	Sand-treatment		Roads	
	Slime-treatment		Ore-treatment plant	
	Tailing dump			

cost a degree of segregation and cost analysis impossible of attainment by cruder methods. The tendency of modern mining practice is to increase the cost analysis both in number and detail. How far it is necessary to go must be decided by the manager and staff. Too minute an analysis of expenditures may serve no useful purpose and may waste time, paper, and storage space.

FIG. 310.—COST OF STOPING

Working Place No.............	Month............, 19.......				Total	Average per ton
Card No.............	Level No.............					
Date started............						
Total tons—est.............	1	2	3	31 spaces		
				Each representing a day		
Labor						
Miners						
Timbermen						
Trammers						
Muckers						
Tool packers						
Miscellaneous						
Supplies						
Timber						
Powder, caps, fuse						
Lubricants and candles						
Miscellaneous						
Power						
Drill shifts						
Drill steel						
Drill sharpening						
Drill repairs						
Air hose						
Air pipe						
Total cost						
Tons ore produced						
Cost per ton						
Cumulative tons						
Average cost per ton						
Tons remaining						

Segregation may be on the basis of time, the working place in the mine, the kind of work, the department, or a broad feature in the operations. The possibilities of segregation are shown in Table 126. In the first column, headed "general," are placed the principal expense items that might apply to any piece of work. The remaining columns give individual operations under mining and development, ore treatment, accessory, repair, and preparation. It is evident that each entry in the last four columns might include all the separate entries in the first column. Under stoping, separate expense accounts may be maintained for each stope, or all the stopes may be grouped together and the expense reported from day to day, weekly, or monthly. In a similar manner each development working may be reported separately, all development workings of the same kind grouped and reported as a group, or all the development workings irrespective of kind grouped together and reported as a whole for the same time periods. In Fig. 310 a form for reporting stoping costs is illustrated. The costs are given for each day and the day's total is the money outgo for working. The total and average cost per ton for each item are given in the last two columns.

A summary form for all stoping operations for a single month is illustrated in Fig. 311.

FIG. 311

Stoping Report for Month............, 19........								
Items	Working Place No.............						Total	Average per ton
Labor								
Supplies								
Power								
Repairs								
Miscellaneous								
Superintendence								
Indirect								
Total cost								
Tons ore produced								
Cost per ton								
Cumulative tons								
Remaining tons								

The system outlined admits of indefinite expansion and is limited only by the degree of differentiation in the primary reports submitted by the timekeeper, storekeeper, shop foremen, engineers' office, mine foremen, shift bosses, bosses, and accountants. At most mines the cost accounts include only the direct outgo, labor, and supplies. Indirect expense, such as management, depreciation, mechanical repairs, etc., are not included, the latter accounts being separately reported each month. They are sometimes apportioned and added to the averages and totals each month. The distribution of power costs is attended with more or less difficulty. The power-plant costs are maintained as a separate

FIG. 312.—ANNUAL REPORT

DETAILS OF MINING EXPENSES, YEAR ENDING_________19__

Supplies	Stopes, tons		Drifts and crosscuts ...... feet		Raises ...... feet		Winzes ...... feet		Diamond drilling ...... feet		Total development ...... feet		Total .... tons	
	Cost	Cost per ton	Cost	Cost per foot	Cost	Cost per foot	Cost	Cost per foot	Cost	Cost per foot	Cost	Cost per foot	Cost	Cost per ton mined
Mine timbers														
Powder														
Caps														
Fuse														
Candles														
Drills and fittings														
Pipe and fittings														
Track and fittings														
Pump and repairs														
Cars and repairs														
Iron and steel														
Change room and office														
Lubricants														
Electric supplies														
Miscellaneous hoisting														
Tools														
Miscellaneous														
Total supplies														
Labor														
Superintendence														
Shift bosses														
Engineers														
Miners														
Muckers														
Timbermen														
Blacksmith and helpers														
Pipe and trackmen														
Cagers														
Pumpmen														
Filling														
Top carmen														
Nippers														
Mine and shaft repairs														
Assay department														
Electrical department														
Engineering department														
Sampling department														
Watchmen														
Surface department														
Diamond drill														
Total labor														
Power–electricity and air														
Totals														

account and the average consumption of power for each division of the work is measured. The unit cost for power production and the average power requirement for each division give the appropriate total power cost to apply.

Monthly, semiannual, and annual cost summaries are made. The amount of detail is greatly condensed, and such reports usually show only the principal groups of expense accounts. Figure 312 gives the annual cost summary used by a large gold mine.

A somewhat condensed summary used by the same mine is given in the following:

TABLE 127

	Cost per ton	Total expense	Percentage of total cost
Stoping .... tons			
Labor			
Supplies			
Power			
Total			
Development.... tons			
Labor			
Supplies			
Power			
Total			
Moving dumps .. tons moved			
Labor			
Supplies			
Power			
Total			
Transportation			
Railroad operation			
Railroad maintenance			
Total			
Milling.......... tons			
Labor			
Supplies			
Power			
Total			
Concentrate treatment.. ...tons			
Labor			
Supplies			
Power			
Total			
Total direct operating costs..........			
Total indirect operating costs			
Total costs............			

The Sales of Product Account.—The transactions are between the mining company and ore-buying or smelting companies. All expense items, sampling, transportation, and treatment are debited against this account.

The Cash Account.—The cash account is a debit and credit account of all bank deposits and withdrawals. Proceeds from ore and product sales are deposited to the credit of the mining company and debited on the cash account. All checks drawn are credited. Checkbook stubs furnish a record of each transaction. A petty cash account is usually necessary and this is handled by depositing a small sum of money at the disposal of the office clerk who is charged with the amount. For each payment he makes out a voucher and at the end of the month turns in the amount represented by the vouchers and receives an equivalent sum, which brings the petty cash fund up to the original amount.

FIG. 313.—PROFIT AND LOSS ACCOUNT

Profit and Loss

- *Income (Earnings)*
  - (1) Ore and concentrates sales less transportation, treatment and charges
  - (2) Bullion sales less marketing expense
  - (3) Earnings from storeroom
  - (4) Earnings from shops
  - (5) Dividends from mine stock of other companies
  - (6) Interest and discounts
  - (7) Rentals and miscellaneous income
- *Outgo (Expense)*
  - *Direct*
    - (8) Mining
      - Labor
      - Supplies
    - (9) Transportation
      - Labor
      - Supplies
    - (10) Ore treatment
      - Labor
      - Supplies
  - *Semidirect*
    - (11) Foremen and shift bosses
    - (12) Development
      - Labor
      - Supplies
    - (13) Pumping
      - Labor
      - Supplies
    - (14) Ventilation
      - Labor
      - Supplies
    - (15) Illumination
      - Labor
      - Supplies
    - (16) Power
      - Labor
      - Supplies
      - Fuel
    - (17) Repair
      - Labor
      - Supplies
    - (18) Timber framing
      - Labor
      - Supplies
    - (19) Tool sharpening
      - Labor
      - Supplies
  - *Indirect*
    - (20) General expense
      - Management staff
      - Office
    - (21) Taxes
      - Bullion
      - Income
      - Property
    - (22) Insurance
    - (23) Compensation
    - (24) Emergency expense
    - (25) Depreciation
    - (26) Miscellaneous

The Profit and Loss Account.—The profit and loss account is a monthly, semiannual, and annual summary of income and outgo. The accompanying chart (Fig. 313) illustrates the principal accounts summarized and one method of classification.

The Balance Sheet.—The balance sheet is an annual statement of assets and liabilities. The accompanying chart (Fig. 314) illustrates the principal items that are included in the statement. Accompanying the balance sheet is often a statement of the ore reserves remaining at the end of the period represented by the balance sheet.

Operating equipment wears out and may be replaced by new machines. A mine plant may also require replacement. A depreciation account is maintained to provide for and to represent this factor adequately as a part of the cost. Some percentage of the installed cost of a machine or the entire plant, buildings, and other appurtenances may be taken, based upon the assumed life of the enterprise. A more detailed account may be carried, in which the life of each piece of equipment and each building or structure is estimated and an annual amount is written off in relation to its installed cost. Necessarily depreciation is not a precise factor, as it is based upon an assumed time that may or may not be accurate. The life of an enterprise can seldom be closely estimated, as changing political and economic conditions have an important influence.

Fig. 314.—Balance Sheet

Balance Sheet

Assets

- (A) Mining properties less amount representing exhaustion of deposit
- (B) Organization expenses
- (C) Mine buildings less total depreciation
- (D) Mine equipment less total depreciation
- (E) Mill buildings less total depreciation
- (F) Mill equipment less total depreciation
- (G) Office buildings and equipment less total depreciation
- (H) Real estate
- (I) Stock owned in other companies
- (J) All prepaid expense
  - Unexpired insurance
  - Unexpired taxes
  - Office supplies
- (K) Stock of mine and mill supplies
- (L) Accounts receivable
- (M) Cash in bank and on hand
- (N) Absorption (mill solutions and pulps)
- (O) Concentrates, bullion, and miscellaneous metallurgical products in process of settlement or sale
- (P) Patents

Liabilities

- (Q) Capital stock authorized less unissued stock
- (R) Accounts payable.
  - Accrued pay roll
  - Accrued expense
- (S) Loans and notes payable
- (T) Net amount realized from operations less amount representing exhaustion of deposit (profit and loss account)
- (U) Miscellaneous
  - Unpaid compensation
  - Unpaid damage suits
  - Unpaid dividends, etc.

Note.—(A) The value of the mining property may be taken as the first cost of the property or may be valued from the profit represented by the ore reserves. The amount written off for depletion is cumulative and is the sum of the annual depletion amounts allowed.

(T) The net amount is the cumulative annual profits represented by the annual profit and loss statements less the cumulative annual depletion amounts.

Improved equipment often renders existing equipment obsolete. Equipment may be replaced by more efficient units long before it has been worn out. Installed cost, less depreciation already written off, plus the cost of replacement parts less their salvage value, enters the cost analysis. The installed cost of the new equipment is subject to depreciation. Obsolescence also applies to buildings and other structures.

The annual rate of mining in a given enterprise may exhaust the mineral deposit within a certain period of years unless additional ore deposits are discovered and developed or other property containing extensions of the deposits is acquired. To provide an accounting basis the value of a deposit is estimated as of a given date, and annual amounts are written off corresponding to the value and to the time period estimated for exhaustion. New discoveries are also valued. The term "depletion" is given to this account. It is an important and controversial subject in respect to income taxes.

Where radical changes are made in the surface plant or new structures are added a separate construction account is maintained. The construction costs are segregated and monthly summaries of expense for labor and materials are prepared.

Where extraordinary expenditures are made or where development is carried greatly in advance of stoping, such expenditures are sometimes carried as suspense accounts. The expenditure is debited to the suspense account and is written off at a regular rate by crediting the suspense account with the amount written off and charging the appropriate regular account with the same amount.

Semidirect accounts include all accounts for which the segregation is not directly apparent but must be determined by special observations, measurements, or arbitrary apportionment. Indirect accounts are accounts which apply to mining operations as a whole and which do not admit of convenient apportionment among the direct accounts. Management, office expenses, surveying, assaying, sampling, prospecting, engineering, accidents, royalties, taxes, insurance, workmen's compensation, hospital, welfare, first-aid work, etc., are items which are generally included under indirect accounts.

Annual Report.—The annual report is a detailed review of operations, a statement of ore reserves, the presentation of annual cost summaries, a profit and loss statement, and a balance sheet. It is prepared by the manager and staff and is for the information of the stockholders. It is usually printed and distributed to the stockholders.

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CHAPTER XVIII

ACCIDENTS AND MINERS' DISEASES

The dangerous nature of mining received early recognition. Agricola in his "De Re Metallica" writes this significant paragraph:

> It remains for me to speak of the ailments and accidents of miners and of the methods by which they can guard against these, for we should always devote more care to maintaining our health, that we may freely perform our bodily function, than to making profits. Of the illnesses some affect the joints, others attack the lungs, some the eyes, and finally some are fatal to man.

The first review of the subject appeared in the sixteenth century. Since then the interest has been intermittent. Great progress has been made in recent decades, and today the subject is of growing importance, not only to the engineer but to the public at large. Active interest is being taken not only in the conservation of natural resources but also in the conservation of men. The work of the state mine inspectors, the Bureau of Mines, the mining companies, and state industrial commissions has made itself felt, and substantial reductions in accident rates have taken place.

STATISTICAL DATA

The fatality rates and injury rates for the years 1934–42, inclusive, are given in Table 128 and causes in Tables 129 and 130.

TABLE 128.—EMPLOYMENT AND ACCIDENT DATA AT METAL AND NONMETAL MINES IN THE UNITED STATES

Year	No. mines	No. men	Avg. days active	Avg. hr. per man-day	Fatal	Injured	Rates	
							Fatal	Injured
1934	3,598	66,645	221	7.89	116	7,892	1.00	67.95
1935	10,819	92,314	220	7.93	164	10,206	1.02	63.27
1936	8,451	100,932	250	8.03	199	14,650	0.98	72.40
1937	9,040	118,429	252	8.02	219	18,055	0.91	75.37
1938	7,233	103,027	228	8.01	156	12,722	0.83	67.61
1939	9,123	111,909	233	7.95	173	13,710	0.84	66.28
1940	11,130	120,120	242	7.94	223	14,766	0.97	64.00
1941	9,620	125,290	255	7.94	230	15,772	0.91	62.17
1942	6,067	112,446	280	7.99	237	13,956	0.94	55.56

NOTE.—Taken from HSS 335, *U.S. Bur. Mines;* rates for fatalities and injured are per million man-hours.

Table 129.—Causes and Numbers of Fatalities and Injuries in All U.S. Mines (Excluding Coal Mines), 1942 and 1914; Injuries for 1942

Causes	Total fatalities, 1942	Total fatalities, 1914	Total injuries, 1942
Underground:			
Fall of rock or ore from roof or wall	70	185	2,307
Rock or ore while loading at working face	1	7	987
Timber or hand tools	...	3	804
Explosives	15	49	84
Haulage accidents	27	31	1,354
Falling down chute, winze, raise, or stope	20	48	852
Run of ore from chute or pocket	7	5	372
Drilling accidents	...	3	844
Electricity	7	8	30
Machinery (not including locomotives or drills)	3	1	465
Mine fires	8	10	5
Suffocation from natural gases	3	1	20
Inrush of water	...	8	1
Stepping on nail	...	...	170
Other causes	9	7	2,507
Total, underground	170	366	10,802
Shaft:			
Falling down shafts	5	34	5
Objects falling down shafts	5	10	61
Breaking of cables	...	2	2
Overwinding	1	...	...
Skip or cage	6	43	75
Other causes	...	11	99
Total, shaft	17	100	242
Surface:			
Mine cars, mine locomotives, or aerial trams	3	3	71
Railway cars and locomotives	1	7	22
Run or fall of ore in or from ore bins	...	5	23
Falls of persons	5	4	186
Stepping on nail	...	...	24
Hand tools, axes, bars, etc	...	...	210
Electricity	1	6	28
Machinery	2	4	216
Other causes	6	8	361
Total, surface	18	37	1,141
Open pit:			
Falls or slides of rock or ore	10	18	108
Explosives	1	11	15
Haulage accidents (locomotives, etc.)	7	9	104
Steam shovels	...	1	49
Falls of persons	3	4	177
Falls of derricks, booms, etc	1	1	13
Run or fall of ore in or from ore bins	...	...	14
Machinery (other than locomotives or steam shovels)	1	5	160
Electricity	...	3	9
Hand tools	...	...	168
Other causes	8	4	458
Total, open pits	31	56	1,275
Grand total	236	559	13,460

TABLE 130.—FATALITIES AND INJURED IN COAL MINES, UNITED STATES

Causes	Fatalities		Injured
	1929	1942	1942
Underground:			
Falls of roof and face	1,182	724	15,890
Mine car and locomotives	413	278	13,447
Gas or dust explosions:			186
Local explosions	195	30	
Major explosions	......	127	
Explosives	88	29	426
Electricity	81	57	1,072
Machinery	28	50	5,631
Shaft	29	7	108
Miscellaneous	45	51	25,811
Total, underground	2,061	1,353	62,571
Stripping or open cut	......	35	1,595
Surface operations	111	83	5,398
Grand total	2,171	1,471	69,564

NOTE.—Injured includes all nonfatal accidents; in 1942 the coal produced per man-hour was 0.703 tons; accidents per million man-hours were 78; accidents per thousand 300-day workers were 165, and average length of shift was 7.05 hr. From *HSS* 333, *U.S. Bur. Mines.*

Fatal accidents in Transvaal gold mines are given in Table 130*a*. In 1940 the average number of workers employed was 42,852 whites and 351,826 colored.

TABLE 130*a*.—FATALITY RATE, TRANSVAAL GOLD MINES
(*Transvaal Chamber of Mines*)

Period	Total fatalities		Death rate per 1000 per annum					
			Underground		Surface		Total	
	White	Colored	White	Colored	White	Colored	White	Colored
1934	57	553	3.42	2.50	0.55	0.80	1.89	2.05
1935	49	658	2.58	2.78	.42	.59	1.42	2.20
1936	60	700	2.65	2.71	.72	.66	1.59	2.16
1937	49	723	2.15	2.78	.37	.61	1.21	2.20
1938	63	667	2.38	2.38	.71	.80	1.46	1.94
1939	50	636	2.04	2.26	.17	.68	1.10	1.83
1940	47	662	1.81	2.08	.35	.80	1.04	1.74

NOTE.—Underground and surface are on average men at work; total is on number employed.

It is impracticable to give more than a limited number of statistical tables. The reference given at the end of the chapter, under the classification "statistical," gives a wealth of information to which the reader is referred for further details. The direct causes enumerated in the various tables are those that result in accidents affecting the individual worker or at most small groups of workers. In coal mines gas, gas and dust, and dust explosions are not of uncommon occurrence and usually result in a large number of fatalities, either as a direct result of the explosion or as a result of the poisonous gases formed. In metal mines fires are the principal cause of accidents affecting a number of workers. The poisonous gases produced by mine fires are responsible for the fatalities.

PREVENTION OF ACCIDENTS

The indirect causes of mine accidents are inherent risk of occupation, defective material, carelessness and recklessness, rate of working, ignorance, disobedience of orders, physical condition of the workers, fault of management, fault of foreman, fault of fellow workers, joint fault of several. To these may well be added deficient illumination of underground workings and inadequate ventilation. Various attempts have been made to determine the relative influence of the indirect causes upon the number of accidents, but these are of questionable accuracy. In the Pennsylvania Annual Report of the Department of Mines it is stated that "If the accidents resulting from carelessness and disobedience of rules could be eliminated, the fatalities in the mines would not be greater than in many of the vocations in the cities." There is a prevailing opinion among mining operators that accidents are due in a large measure to carelessness and ignorance, but this is denied by some. However that may be, the fact remains that a large number of accidents are preventable, and since accidents, not only from a humanitarian viewpoint but also from financial considerations, are to be avoided, the management of a mine must give the subject an important place in its activities. Accident prevention is a subtle problem and requires intensive thought and something more than the placing of "safety first" signs in conspicuous places and the introduction of safety devices. Each mine is a problem peculiar to itself not only in respect to the physical conditions but also with respect to the workers. It is a never-ending problem during the life of a given mine.

A study of accidents which have occurred, with the object of preventing a repetition, and a review of the indirect causes, with the object of eliminating as many as possible, are two necessary preliminaries. In the following are discussed the various methods of accident prevention which are characteristic of mining practice in different localities or which are worthy of consideration as an approach to the solution of the problem.

Selection of Operating Officials and Foremen.—The selection of the superintendent is of first importance. An experienced, careful superintendent is an insurance that similar men will be selected for the under positions. The superintendent, his foreman and shift bosses are on the firing line. It is their duty not only to get the ore out but also to get it with a minimum of risk to themselves and the workers. They are partly responsible for the selection of methods and wholly responsible for the execution of the work. Methods and tasks should be systematized as much as the conditions permit. Regulations should be drawn up for the direction of the workers in each division, and these should be posted and copies given to each new worker. Each worker should be required to be familiar with his working directions and held responsible for their observance. The superintendent and foremen should hold frequent conferences at which should be discussed not only the usual features of the work but also the risk factor. Methods and procedure should be determined by considerations of safety first and costs last. Where the work is dangerous it should not be pushed, and the workers should be allowed to take their own time.

Selection of Workers.—The selection of the workers is of next importance. It is not usually possible to select a crew of several hundred miners and other workers without getting a number of inexperienced and incompetent men. These should be gradually weeded out. Where a good but inexperienced man is found he should be paired off with an experienced worker and allowed time in which to develop. Green men should always be placed in working places with men familiar with the mine. A certain amount of instruction and training work should be expected of the management. A large mine should maintain an instruction and trying-out squad. This should be placed in the charge of an all-round experienced miner, whose duty should be to instruct the new men in the mine work and acquaint them with any peculiar and dangerous conditions. Each man should be given a fair trial, and only those who qualify in a reasonable time should be retained. An *esprit de corps* should be cultivated among the men. This can be done by the recognition of good work. A small bonus payment may go a long way toward developing a spirit of cooperation between the workers and the management.

As it takes time and money to build up a working force, it is poor business policy to let good workers leave. Good living conditions and fair treatment will in most cases prevent numerous changes. A stable working force greatly simplifies accident prevention. One of the important factors operating in foreign countries to reduce accident and fatality ratios is the availability of a mining population which does not shift about and which has its mining experience as a heritage from present and past generations. While some mining centers in the United States are characterized by a relatively stable and experienced mining population, many have to contend with recruiting labor from many localities. The present need is for experienced miners.

Training and Education of Workers.—Provision in our educational system for the training and education of miners is in its beginning stages. It is a community problem rather than one that concerns the mining company, although most progressively managed mining companies recognize the importance of the work and frequently lend financial assistance to further it. Night schools should be established in every mining center, and where the miners are foreigners instruction in the English language should be an important part of the curriculum. Elementary mining, such as the use of explosives and the causes and the prevention of accidents, as well as first aid should be presented in a simple, practical manner. Charts, lantern slides, and illustrative material are essential in imparting knowledge of this kind. Alertness of mind is an important factor in preventing accidents. The humdrum of vocational work has without doubt a strong tendency to produce dullness, which can be overcome in part at least by the stimulation of interest in other things.

First-aid contests that take on the features of a game serve two important purposes, instruction and training in first aid and accident prevention, and the quickening of mental responses. The miners' classwork under the leadership of the Young Men's Christian Association in certain mining localities serves a similar purpose and is to be commended.

Secondary schools for the training of mine foremen should be a part of the educational system in centers where large numbers of miners are employed. The course of study should include all the subjects enumerated before and in addition elementary chemistry, mathematics, surveying, and accounting. The study of mining machinery and methods should be especially thorough. The course of study should be open to miners who have shown special aptitude and ability.

Discipline of Workers.—Obedience to orders by the workers is essential in the operation of a mine where dangerous conditions prevail. A sufficient number of foremen, shift bosses, and bosses is necessary in order to cover the working places adequately. They should be required to report all infractions of regulations as well as carelessness or incompetence on the part of the workers. The breaking of a safety regulation should be brought to the immediate attention of the worker and he should be warned. A repetition of the offense should be met by a temporary layoff and a third infraction by discharge. The thoroughness of the supervision of the workers is a factor in developing a well-disciplined mine force.

Safety Regulations.—A general set of safety regulations should be compiled for the guidance of workers at each mine. The regulations should be prepared by the mine staff and foremen and should apply to the surface plant and underground workings. They should be segregated into groups, such as those pertaining to the handling of explosives, hoisting, tramming, raising, shaft work, stoping, drifting, and shop regulations. Copies should be posted about the surface plant and each worker should be provided with a copy. The Cleveland-Cliffs Iron Company issued at one time a safety-regulation book in English, Finnish, and Italian and gave a copy to each employee, requiring him to sign an agreement to study and live up to the regulations. The employee must satisfy his foreman or boss within a reasonable time that he is familiar with the regulations. The same company posts copies of bell-signals and special rules governing the use of explosives and the duties of hoisting engineers and motormen.

Many examples of safety rules in use by different mining companies have been published and are available in technical publications. While some of these regulations would apply to any mine, it is evident that safety regulations must be drawn up with particular reference to the mine in which they are to be used. The underground safety regulations of the Copper Queen mine (Arizona) are given as an example.

General rules for the prevention of accidents at the Copper Queen Company's mines, in Arizona, include the following for underground work: 1. As all mining work is hazardous, extra care should be taken not only for your own safety, but for the safety of men working with you. 2. Watch for danger signals; they are often unnoticed if there is not sufficient light. 3. Every manhole and place of refuge shall be kept constantly clear, and no refuse shall be placed therein, and no person shall in any way prevent access thereto. 4. The general condition of the timbering in the mine shall be safe. The men shall take all the necessary precautions to ensure the safety of the timber in the working places. 5. In all stopes where square sets are used, it shall be the duty of men working in stopes to see that the floors are properly centered on the caps, particularly after blasting, and spiked wherever deemed necessary 6. When working in heavy or untimbered ground, care should be taken that there is plenty of room for a quick exit. All obstructions such as cars, wheelbarrows, etc., should be removed out of the way. 7. Existing winzes opening directly from the floor of the drift or stope must be kept covered by a substantial hatch or planking except when in use, at which time the passage to persons other than those working at the winze shall be barred off by a substantial rail across the roads of access to the openings. 8. The miners shall be responsible for the safety of the roof and walls of their working places. 9. In the mining, care must be taken in approaching workings thought to be filled with water, and the boreholes must be kept at least 20 ft. in advance of the drive. 10. Planking over sumps and ditches must be kept secure. 11. No candle or lamp shall be left burning in a mine when the person using the candle or lamp departs from his work for the day. Sconces must positively be used, except when candlesticks or carbide lamps are employed. Lights must be placed so that timber cannot catch fire. 12. Employees shall, or soon as it is discovered, inform the foreman or shift boss of the unsafe condition of any working place. 13. Be sure that chutes are protected so that men cannot fall into them. 14.

Men are strictly forbidden to carry tools upon their shoulders in any drifts where electric wires are installed. It is very dangerous and may result in death. 15. Two openings to the surface are provided by law, except in the case of mines that are being opened. 16. Men should learn the different openings from their place of work, and their attention is called to signs at the intersection of drifts which direct them to shafts or outlets.

Safety Devices.—Gates at stations and landing platforms in shafts, doors at manways, gratings at chutes, and covers over shaft and winze openings not in use should be provided. Ladderways should receive special attention, and platforms used more than occasionally should be provided with handrails at dangerous points. Switchboards and transformers should be protected by a wooden railing. Machinery should be protected by railing. Gears should be enclosed and belt runs railed off. Underground trolley wires should be placed at such a height as to prevent any possibility of contact, and where workers have to pass under them the trolley wire should be protected by wooden boxes. Electric cables in drifts, adits, and shafts should be thoroughly insulated and suspended from insulators in such positions as to preclude the possibility of interference.

Hoists should be provided with overwinding and excessive speed stops that should operate automatically. Where these are not provided, detaching hooks should be used. Cages should be provided with gates and safety catches.

In the surface plant all gears and belt runs should be protected by casing or railing. Guards should be placed on all circular saws and planing machines. Wherever workers have to pass close to machines in operation, a housing should be placed about the moving parts of the machine. Accidents can be prevented by the systematic inspection of the mechanical equipment while in operation and the devising of housing or rail protection as required.

Among the minor though important safety devices are goggles and respirators. Samplers and miners engaged at tasks where rock chips are liable to be thrown should be required to wear heavy glass goggles. They are also necessary in machine shops when castings are being chipped or emery wheels are being operated. Respirators should be worn where workers have tasks in dust-laden air. Sample grinding, dry-crushing plants, chute loading, and rock drilling in raises and stopes necessitate the use of respirators except where special precautions are taken to eliminate the dust.

The use of hard hats by miners engaged in shaft sinking and underhand stoping as a safeguard against falling pieces of rock is now widespread. Hard-toed shoes, leg- and foot-guards, and heavy gloves also reduce the number of minor accidents.

Design of Underground and Surface Plant.—Much can be accomplished by the engineer in designing the underground layout and equipment of the mine. Top and side clearance limits should be established for adits, drifts, and crosscuts. Car and locomotive clearances should be standardized in a similar manner. Underground machinery installations should also have certain limiting side and top clearances. Particular care should be taken in designing the details of illumination. All places where machinery is in operation should be especially well illuminated. Crossings of traveling and haulageways, stations, and landing platforms require good illumination. Refuge chambers, exits, traveling and haulageways require particular attention and the separation of haulage- and traveling ways should be provided for wherever possible. Ladderways should be carefully designed and lighted. For locomotive haulageways a simple system of block signals should be installed to preclude the possibility of collisions and to warn track repairers of the presence of trains. Chutes, chute gates, and skip loading pockets should be of such design that loaders are amply protected from possibility of injury. The size of all parts and the selection of the material should be such that an ample margin of safety is obtained in all underground structures.

In the surface plant the track, machinery, and head clearances should be carefully considered. Machinery spacing should be such that the risk involved in cleaning and lubrication is reduced to a minimum. Foot planks and stairways should be designed of ample strength and provided with pipe or wooden railing. Shafting and pulleys should be placed so that workers cannot come in contact with them. Gears should be housed and belt and rope runs protected up to at least 4 ft. above floors. Where haulage is a feature of the surface plant, separate footways and over- or undercrossings should be provided and haulage tracks fenced off. Fire protection of both surface plant and underground workings should be provided for in the initial plant and development plans.

Warning Signs.—Safety regulations are supplemented by warning signs placed at critical places. These signs should convey a specific warning and should be illuminated, to enable them to be read at a distance. Magazines and crossings of traveling and haulageways, chute openings, and other dangerous points should be marked. Direction signs indicating the direction to connecting manways and exit shafts are necessary.

Safety Inspection.—Safety inspection and supervision play their part in the prevention of accidents where adequate provision is made for them. In a small mine, the superintendent and foremen must of necessity perform this duty, but in a large mine a special inspector should be employed. A sufficient interval should be allowed between shifts to enable him to make the rounds and determine the condition of the working places. In most coal-mining states the safety inspector is the fire boss. His usual duty is to test rooms and entries for gas. To this might well be added the examination of the roof, although this duty is more often placed in charge of the timber boss. In heavily timbered metal mines a fire boss usually inspects all stopes after the miners leave. At most mines, whether small or large, the working shafts are thoroughly inspected at least once each week, or every day where they are in bad ground. Hoisting ropes are inspected each day by a competent ropeman. The hoist should be inspected by each engineer before he assumes charge at the beginning of the shift. Workers coming off shift report missed holes and any dangerous conditions in the stopes to the shift boss of the oncoming shift. The usual practice is to bulletin such information on boards at the stations or at the shaft collar. The foreman and his shift boss make at least one round of inspection during their shift and give specific directions to the workers.

The Cleveland Cliffs Iron Company employs a safety inspector, who is placed in charge of the safety department. His duty is to inspect all places where men are working or through which they are obliged to travel. He reports the failure to observe safety regulations and all dangerous conditions directly to the company's agent, to whom alone he is responsible. His inspection visits are unannounced and serve as a check upon the men in charge of the mine. His report comprises a series of answers to definite questions, two groups of which are here given as an example.

SHAFTS:

1. Is protection at collar of shaft sufficient and in good order? (Rule 51)
2. Is opening to shaft at timber tunnel properly protected and in good order?
3. Is protection at shaft stations sufficient and in good order?
4. Are there skip tenders and at what levels? (Rule 7)
5. Is there a cage rider?
6. What tools are allowed with men riding in skip, cage, or bucket? (Rule 8)
7. Are projecting tools properly lashed to hoisting ropes? (Rule 9)
8. How often are timbers in manways cleaned? (Rule 22)
9. Are they in good condition?
10. Are working stations sufficiently lighted? (Sec. 26)
11. Are there passageways at all levels around hoisting compartment? (Rule 44)
12. Are they in good condition?
13. Are there guides for the bucket?
14. What style of crosshead is used? (Rule 15)
15. What clearance is provided on guides?
16. Is stopper securely fastened to hoisting rope at least 7 ft. above rim of bucket? (Rule 15)
17. Are guides and crosshead kept in good condition?

18. Is there more than one outlet to surface?
19. Are there connections between levels other than the main shaft? (Sec. 38)
20. Is the second outlet kept in good condition? (Sec. 38)
21. Is condition of pipes, electric wires, and conduits safe?
22. Are steam pipes covered or protected from accidental contact?
23. Is the general condition of the compartment through which men are hoisted safe for men as regards lagging, timber, guides, etc.?

General Mining:

1. Is general condition of timbering or other means of support throughout the mine satisfactory? (Sec. 12)
2. Are mine maps clear and accurate for purpose of inspection? (Sec. 22)
3. Are all dangerous places fenced off? (Rule 4)
4. Are proper danger signal boards displayed at all dangerous places? (Rule 4)
5. Are candles or lamps left burning after shift? (Rule 5)
6. Are sumps securely planked over? (Rule 45)
7. In passageways are roofs and walls securely lagged? (Rule 46)
8. Are winzes and raises in direct line of drift? (Rule 47)
9. Are winzes, raises, and open stopes properly guarded? (Rule 48)
10. Are ladderways in winzes and raises located in drifts properly protected by hatches? (Rule 49)
11. Are all chutes in winzes and raises properly protected by gratings?
12. Is proper provision made for safety of men working at chutes?
13. Are the communications between contiguous mines in good condition? (Sec. 39)
14. What is the condition of ventilation in different parts of the mine? (Sec. 43)
15. Are sufficient dry closets maintained? (Sec. 44)
16. While the responsibility for the safety of the roof and walls in the individual places is upon the workman, are they also inspected by the superintendent, captain, and shift bosses, and how often? (Sec. 45)
17. Are the rules for safety pillars on boundaries observed? (Sec. 46)
18. In approaching workings known to be flooded are proper precautions taken? (Sec. 37, Rules 32–33–34)
19. Complaints from workmen—report, if any.

In addition, a foremen's safety committee, consisting of three members, makes periodic inspections in accordance with certain instructions issued by the company's agent. The safety inspector accompanies the committee on their tour of inspection in the capacity of secretary. He has no voice in the committee's decisions. He prepares the report, which is reviewed and signed by the committee. A workmen's safety committee cooperates with the safety inspector and foremen's committee and approves or recommends changes in the working conditions of each mine. A central safety committee, consisting of the company's mine superintendents, the head mining captain, the master mechanic, the secretary of the pension department, and the safety inspector, with the company's agent as exofficio member, meets monthly and acts as a legislative body on all safety recommendations received from the foremen's and workmen's committees. A majority vote is recorded and reported to the company's agent, and the recommendation, when approved, is put in force. The decision on all rules and regulations rests with the central committee. All accidents, whether fatal, serious, or slight, are reported to the central committee and carefully considered. Recommendations are made for avoiding the recurrence of preventable accidents.[1]

The Anaconda Copper Mining Company maintains a safety organization consisting of a bureau of safety, which includes all the managers of divisions, a general safety committee, which includes the general manager, assistant general manager, and general superintendents, and a safety committee for each group of mines. Separate safety committees control at the reduction plants.

The two safety organizations are exemplifications of the requirements of large mining companies and would be inapplicable to small companies. For such companies two committees, one including the superintendent, foremen, and shift bosses and the other selected from the workers, are all that would be necessary for the control of the inspection work.

Annual, semiannual, or more frequent inspections are made by state mine or county inspectors where such officials are provided for in the governmental organization. Most state laws require an inspection of every mine at least once a year and the investigation of each fatal accident by the inspector or deputy inspectors. Under the state law the mine inspector is required to see that the laws pertaining to safety in mines are obeyed.

Legislation.—Legislation concerning safety regulations has been enacted in many mining states. In coal-mining states quite complete mining codes are usually in force. In metal-mining states the legislative enactments affecting the operation of metal mines are not so complete; in fact, they are quite fragmentary as a rule and sometimes poorly enforced. In some states the office of mine inspector is elective and in others appointive. In some the inspector is given discretionary power, while in others specific regulations are drafted into the law and he is charged with their enforcement.

Safety Bonus and Safety Records.—Human inertia may tax the ingenuity of a manager in his efforts to make effective a well-planned system of accident prevention. How inertia can be overcome is illustrated by the methods in use by the New Jersey Zinc Company, which were described by B. F. Tillson at a meeting of the New York section of the Mining and Metallurgical Society of America. The company adopted the plan of offering a $200 prize at the end of the year to the mine boss or timber boss having the best record for freedom from serious accident. Subsequently the prize system was changed to a bonus system.

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[1] System of Safety Inspection of the Cleveland Cliffs Iron Co. William Conibear, *Trans. Lake Superior Min. Inst.*, vol. 17, p. 94.

The Anaconda company adopted an accident-prevention bonus system in 1935 for foremen and shift bosses based on accident experience but noncompetitive in character (see the bibliography). At the Union Pacific Coal Company a prize system was adopted, with competition by all underground employees (exclusive of foremen) who passed through the year without a lost-time accident; subsequently monthly cash prizes were given in addition to the annual grand prize. Speaking of this program, Eugene McAuliffe states that men are respondent to patient educative effort and a continuing appeal to their better side, rather than to wholesale disciplinary measures, and that some novelty that will engage and hold their attention must be employed if results are to be obtained in any effort as intangible as mine safety. A combination of persistent, patient educational effort, plus willingness of the management to assume full responsibility of leadership, plus a further attraction of material reward, will ultimately show results (see the bibliography).

D. E. Woodbridge[1] pithily summarizes the whole subject of mine accidents and accident prevention in the following.

> In the pursuit of reform in conditions at mines, as elsewhere, there is the danger of going too far. The extremist for profit will view mine-safety expenditures with an eye to immediate costs of installation and operation, whereas the idealist will urge such a refinement of methods, such a thorough protection, and so wide-reaching a paternalism as will be unjust and will instill too great confidence in the minds of operators, and therefore produce carelessness. It is unnecessary to argue that there are inherent and unavoidable risks in industry. The fact is recognized by all except a very few unreasoning enthusiasts. Any system of safeguarding employees that minimizes their sense of personal responsibility tends to increase casualties. Many employers hold that safety precautions may be carried so far that this sense of personal obligation will not only be diminished, but that it will be entirely destroyed. The responsibility for preventable accidents lies not with the employer alone, but with the employee as well. Both usually suffer through careless acts. No sentimental considerations, no effort to take a position where there can be no criticism of his zeal for safety, should cause one to forget that both parties are accountable and that they have duties equally important and urgent.

COAL-MINE EXPLOSIONS

Cause and Nature of Explosions.—Explosions are produced by the ignition of mixtures of marsh gas and air, marsh gas, air, and coal dust in suspension, and air and coal dust in suspension. Ignition may be caused by an open lamp, the opening of a safety lamp, the use of a defective safety lamp, the lighting of a match, arcing from electrical switches, trolley wires, and electrical machinery, or a blown-out shot caused by the use of heavy charges of black powder, careless tamping, or both.

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[1] *T.P.* 30, *U.S. Bur. Mines*, p. 36.

Mixtures of methane and air may be purely local, and when ignition takes place the explosion rapidly propagates itself throughout the zone occupied by the explosive mixture. While gas explosions are usually local, they are liable by the stirring up of fine coal dust to spread throughout a mine.

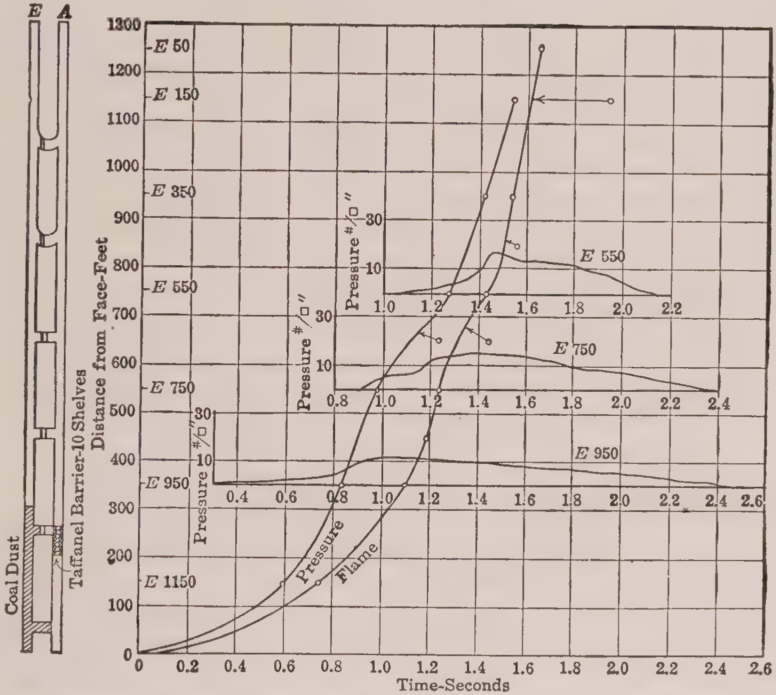

FIG. 315.—Diagram of temperature, pressure and flame meaurements taken in a coal dust explosion test. (*U.S. Bureau of Mines.*)

G. S. Rice, describing the action of a dust explosion, states that where the ignition is produced by an explosion of black powder an initial "shock wave" or "pioneering wave" is started, which travels at the rate of a sound wave. Tests have shown an average velocity of 1,150 ft. per sec. The pressure produced by the shock wave increases up to 3 or 4 lb. and decreases in intensity, though maintaining its velocity, as it progresses from the point of ignition. Dust is raised by the shock wave and burns rapidly. There is an interval of about half a second before the pressure produced by the rapidly burning dust becomes noticeable. The explosion flame takes from ½ to 1 sec. to traverse the first 100 ft. and then rapidly increases in velocity until velocities of 2,000 ft. or more per second are attained. The pressure rapidly rises as the explosion travels, reaching as much as 73 lb. per sq. in. at a distance of 350 ft. from the origin; 119 lb. per sq. in. at 750 ft. has been reached. Where coal dust is in suspension the explosion propagates itself throughout the dust zone and farther in most instances, on account of the "pioneering wave," which raises additional dust as it travels. Figure 315 illustrates the pressure, flame, and velocity observations made of an explosion in the experimental mine of the Bureau of Mines. Rice states that coal dust in the proportion of 0.032 oz. per cu. ft. was ignited, and Taffanel reports an ignition with a proportion as low as 0.023 oz. per cu. ft.[1]

The prevention of coal-mine explosions has received the earnest attention of coal-mining engineers in all coal-mining countries. England, France, Belgium, and Germany have encouraged experimental investigation of the subject. Since 1907 the work of investigation in the United States has rapidly progressed and valuable conclusions have been reached by the engineers and scientists of the Bureau of Mines.

Prevention of Mine Explosions.—Gas explosions are prevented by thorough ventilation and by removing the causes of ignition. In gaseous mines all open lights are forbidden and only approved forms of safety lights used. Trolley locomotives are allowed only on the intake airways. No uninsulated conductors are permitted in rooms or gaseous portions of the mine, and the motors and electrical switches used in such parts must be completely enclosed in explosion-proof enclosures of noninflammable material. Electrical equipment is now of the permissive type, which signifies that it has been thoroughly tested and has been approved by the Bureau of Mines for use in gaseous mines. Electrical coal-cutting machines must be operated by miners competent to test for the presence of gas, and such machines must not be operated for longer periods than half an hour without an examination for gas. Permissible explosives in charges not more than 1.5 lb. should be used for blasting. Ignition of blasts is effected by means of an electric current, which is switched on to the leads from a position outside the mine after the miners have completed their shift and have left the mine. Where excessive quantities of gas are discharged in any part of a mine, working is discontinued in that part. All workings are tested by fire bosses before workers are allowed to enter them. Frequent tests on the air coming from splits and the main return airway are made and the quantity of methane present is determined. Old workings are either completely sealed off or thoroughly ventilated. Doors and stoppings are carefully constructed and their tightness is tested from time to time. Air currents are measured at regular intervals and the proper quantity of air is adjusted in accordance with the measurements. Pockets of gas are removed by bratticing as soon as detected. A recording barometer should be installed at each gaseous mine and attention should be given to its indications. It has been shown that in some mines a low barometer is coincident with an increased flow of gas into the workings.

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[1] Investigations of Coal Dust Explosions. G. S. Rice, *Trans. A.I.M.E.*, vol. 50, p. 552. See also *Bull.* 167, *U.S. Bur. Mines.*

Coal dust varies in its susceptibility to ignition when suspended in air. It is necessary, therefore, to test the coal in a given mine and to determine whether a dangerous dust is formed. Clean coals, containing a large proportion of volatile combustible, are especially dangerous. Preventive measures are the removal of the dust at regular intervals, the wetting down of the accumulated dust, the prevention of its general distribution in the workings, and the elimination of all causes of ignition.

Dry, dusty mines are especially dangerous. The winter season is more critical, since the ventilating currents tend to dry the air. Steam and water sprays are necessary to charge incoming air currents with sufficient water to prevent the drying out of the mine. While it is possible to remove the dust from the haulageways, it is also necessary to wet it down thoroughly at frequent intervals. The coal dust should at all times be sufficiently moist to ball up on the hand. Sprinkling cars are used to distribute the water; in some mines a water-pipe system is extended throughout the mine and the water is distributed from hose. Ribs and timbers are also washed down. The general distribution of the dust along roadways is prevented by using tight metal cars for haulage. End-dumping wooden cars should never be used in a dusty mine. Calcium chloride in a solution ranging from 2 to 6 per cent has been found efficient in some mines. Its use obviates the necessity of of frequent wetting down, but ribs and timbers must be cleaned. To prevent ignition of coal dust, only permissible explosives must be used; coal must be undercut and not broken "off the solid"; holes must not be drilled deeper than the undercutting or shearing; no more than two shots should be fired simultaneously; where shots are fired from a station outside the mine, the sequence of the blasts should be arranged so that the first are on the outtake air and the blasts progress successively toward the intake airway; holes should not be tamped with coal dust or small coal.[1] The precautions enumerated under gas explosions apply to dusty mines as well.

The periodical sprinkling of generous quantities of rock dust has the effect of rendering the coal dust less susceptible to explosion. G. S. Rice states that dry dustless zones do not stop or check an explosion; he advocates the wetting of the dust throughout the mine or the use of inert dust. He notes that the flame of an explosion has penetrated dampened zones 500 to 600 ft. in length. To prevent the propagation of explosions beyond a limited zone, Taffanel made use of rock-dust barriers. These consist of a series of shelves heavily charged with rock dust. They are placed at distances of 2 yd. apart, a shelf 20 in. wide being used. A barrier consists of 15 shelves placed across a roadway. The shelves are tripped by the initial wave of the explosion and the inert dust is discharged into the air. The dust no doubt acts by cooling the flame temperature to a point below that of ignition. Rock-dust barriers have been found to be successful in stopping explosions. It is now a common practice to rock-dust all workings at intervals, except working coal faces, with a nonsiliceous dust. At working places coal dust is kept down by watering coal cuttings and coal piles. Empty cars are wetted down before being returned for loading. Frequent tests are made to determine whether coal dust accumulates in dangerous proportion.

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[1] *Miners' Circ. 3, U.S. Bur. Mines.*

FIRES IN METALLIFEROUS MINES

Most fires in metalliferous mines are preventable. The causes are the ignition of inflammable material such as timber, oils, and waste; carelessness with candles, lamps, and smoking; smoldering fuse resulting from a blast; overheated bearings on machinery; short-circuiting and overheating of electric wires; and spontaneous combustion.

The methods of prevention are as follows: A close control of all inflammable material on the surface or underground is maintained. The amounts that are permitted underground are prescribed and their proper storage is provided for. Regulations covering smoking and the use of candles and lamps are drawn up and enforced by the foremen and shift bosses. A system of fire inspection is installed in all heavily timbered mines. Hand fire extinguishers are placed at critical points, and a water system and hose reels are installed at stations and heavily timbered stopes. A fire-fighting squad is formed and the men are trained in the handling of the fire apparatus. The possibility of fire breaking out in different parts of the mine and the surface plant is considered and methods of control are planned. Steel fire doors are placed on levels so as to aid in isolating a possible fire. A system of alarm signals for surface and underground use is installed. Stench warnings (ethyl mercaptan or amyl acetate introduced into the compressed-air pipe system) are effective, though they are not suitable for use in air currents.

When an underground fire has started, the first step is to get all the men out as speedily as possible. Measures are then taken to fight the fire. These consist in bringing hose lines to the fire and endeavoring to extinguish it directly by the application of water or, where this is not practicable, by constructing bulkheads to prevent the fire from spreading and to cut off its supply of air. Oxygen apparatus has made possible the more or less complete bulkheading of a fire. Without such apparatus bulkheads can be constructed only on the incoming air. Where control cannot be obtained in this manner the mine openings are sealed and the fire is smothered. Steam is sometimes turned into the workings with the same object. Flooding the mine is usually the last resort.

Entering a mine in which a fire has been extinguished by sealing is attended with considerable risk, as the workings are charged with carbon dioxide and carbon monoxide. The oxygen apparatus should be used under these conditions.

One of the great dangers in mine fires is that the men may be trapped in the mine. State laws require that every mine maintain at least two exists. In the Argonaut mine (California) two exits had been provided in accordance with the existing law. A fire occurred in the main working shaft, a downcast, at the 3,000-ft. level, while some 47 miners were at work upon the lower levels of the mine. The upcast was connected with an exhaust fan, the top of the shaft being closed with a stopping. This upcast shaft served as the second exit and was provided with a ladderway. Raises connected the lower levels with the upcast. The downward air current in the main shaft under the influence of the fan quickly filled the lower workings and the upcast with the gases from the fire. The trapped miners proceeded toward the upcast but were unable to get there and bulkheaded themselves in, all perishing from the poisonous gases. Caves in the main shaft broke the compressed-air pipe, telephone, and signal lines. Here, obviously, the second exit was insufficient. Mining practice has evolved several possible methods of meeting a situation of this kind: The maintenance of connections with neighboring mines at critical levels; the construction of refuge chambers of gas- and fireproof construction and provided with a water supply and a compressed-air line independent of the ordinary air mains; and a third exit. In the place of one or more of these expedients careful consideration should be given to the equipment of dry timbered shafts with an automatic or open-sprinkler system or the installation of an adequate fire-fighting system in the critical places in the shaft. Another alternative is the guniting of all shaft and station timbers. This has been tried and has proved successful for a considerable period, but dry rot in many cases so weakened the timbers that guniting was discarded for such fireproofing.

Another critical situation arising in mine fires is the effect of the fire upon ventilating currents. Where the ventilation of a mine is natural, the occurrence of a fire in the upcast merely accelerates the ventilating current in the same direction. If it occurs in the downcast there is a tendency to reverse the direction of the downcast air current. If the ventilating head is too pronounced, the downcast air current will not be reversed and the gaseous products of the fire will be drawn downward. There is much uncertainty as to what will result under natural ventilation conditions where a fire occurs within the mine away from either inlet or outlet. The natural tendency is for the heated gases to rise wherever possible, as through connecting raises and stopes, and to follow the outgoing air current. However, the volume of heated gases may be such that less resistance is encountered through the inlet passage, the result being a reversal of the air current. As a consequence of this uncertainty there has arisen in metal-mining practice a reluctance to do anything that would change the direction of the air currents after a fire has started underground. The turning of water down an upcast, for example, has been known to reverse the ventilating current.

Under controlled ventilation the conditions are quite different. In practically all cases the established direction of the ventilation determines the spread of the gases, for whereas a fire burns slowly against the ventilating current, it spreads and burns rapidly in the direction of the ventilating current. In mining practice there has arisen a tradition that has been formulated into a dictum: "Never reverse the ventilating current." By a little thought the absurdity of holding tenaciously to this dictum becomes apparent. If the fire occurs on the outlet leg of the ventilating system, the zone between the fire and the outlet becomes filled with the gases produced by the combustion; that is, it becomes the dangerous zone. Clearly, to reverse the direction of the air current under such conditions would spread the gases in the opposite direction all the way from the fire to the inlet. On the other hand, if the fire occurs on the inlet side of the ventilating system, the danger zone extends to the outlet of the mine. If the fan is shut off promptly, there is a strong probability that the heated gases will rise and escape through the inlet; in the opposite direction the resistance is greatest, and it is unlikely that the gases would seek the line of greatest resistance. By the prompt reversal of the direction of the air current, the same conditions would be set up, only in greater degree. This is a specific case where the fan should at least be stopped, or, better, reversed. If the fire occurs in the heart of the mine (assuming controlled ventilation) the half of the mine between the fire and the upcast (outlet) would be filled with gas, while the half toward the inlet would be safe. Dead spaces on the fire side would be neutral, but diffusion of gases would eventually take place and pollute the stagnant zone. In this case there is nothing to be gained either by stoppage or reversal of the ventilation. In fact, with stoppage the gases would diffuse in all directions, and with reversal they would be driven back into the only safe part of the mine.

What should be done to control ventilating currents when a fire starts? The important controlling factor is the situation of the fire in the mine.

The procedure then must result from an intelligent analysis of the situation and prompt action to keep as large a portion of the mine as possible comparatively safe and freed from gases.

FIRST AID AND RESCUE WORK

First Aid.—The term is used to signify the assistance rendered a victim immediately after an accident and before competent medical assistance has arrived. While certain injuries are apparent and proper assistance can be rendered by persons who have received first-aid training and are accustomed to act in emergencies, there are other injuries that are not so readily diagnosed. Usually little harm can result from the application of bandages and splints, although there is always the possible danger of further injury to the victim. Where the nature of the injury cannot be determined by a superficial examination, it is a safe rule to treat the accident as serious and to handle the patient with the utmost care. An accident in a stope or other working place necessitates the removal of the patient to the shaft station. If he is unable to handle himself, a stretcher of such a type as to admit of being lowered easily in a manway or raise should be secured and used in removing him from the working chamber. Stretchers of this kind differ from the ordinary military type in that they are constructed of wire and are basket-shaped. Straps for securely fastening the patient are provided. A ring in the head end enables a rope to be attached for lowering through the manway. Each important station should be equipped with a stretcher of this type and a first-aid kit.

At most mines the present practice is to train a certain number of the workers as first-aid men, who are called upon to render assistance when an accident occurs. The superintendent, foremen, shift bosses, and bosses should be required to take the first-aid training. It is necessary for them to assume charge and direct the work of removing the victims of an accident.

At the surface a special room is set aside as an emergency hospital. It should be equipped with table, stretchers, hot and cold water, and a complete outfit of bandages and surgical essentials. A thoroughly clean, light, and airy room is desirable. It should be used for no other purpose.

The details of first-aid training for miners are given in the Miners' Edition of the American Red Cross "Abridged Textbook of First Aid." The training squad is usually instructed by the company's physician and meets at regular intervals.

Mine Rescue Work and Rescue Apparatus.—The rescuing of miners caught in a mine fire or trapped in workings by an explosion necessitates the use of apparatus that will enable the rescuers to work in irrespirable or poisonous gases. Two forms of apparatus are available for work of this kind. The first consists of a smoke helmet to which is attached a length of air hose, through which air is pumped to the wearer by a bellows placed in such a position as to command a supply of pure air. Apparatus of this kind is limited in linear range. By the use of compressed air a greater length of hose can be used, and it is probable that a distance of 500 ft. or more could be attained. Heise and Herbst give a range distance of 600 ft. A speaking-tube attachment or a telephone is a desirable addition. The second form of apparatus is self-contained and comprises a supply of oxygen and a regenerator which removes the carbon dioxide exhaled from the supply, circulating in a closed circuit. The supply of oxygen is sufficient for from 1 to 2 hr. use.

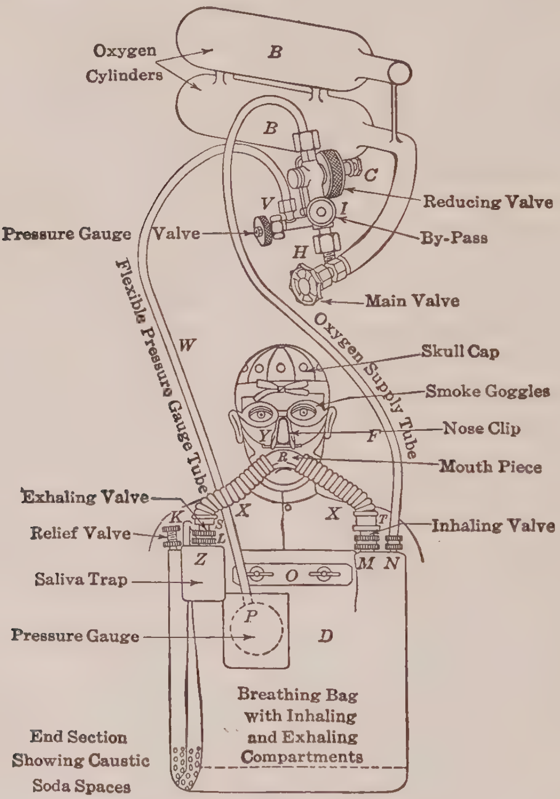

Fig. 316.—Proto oxygen apparatus.

Figure 316 is a diagrammatic view of the Proto apparatus. The oxygen supply is contained in a pair of steel cylinders, each of which contains about 5 cu. ft. of oxygen compressed to 120 atmospheres or 1,800 lb. per sq. in. The main valve *H* controls the supply. The oxygen passes through a reducing valve *C*, which regulates the flow so as to pass from 1½ to 2 liters per min. The oxygen supply passes from the reducing valve to the breathing bag through the tube *F*. The breathing bag is constructed of rubber and is divided into two compartments by a rubber partition. The bag also serves as the regenerator, and for this purpose contains a 4-lb. charge of stick caustic soda. The caustic soda combines with the carbon dioxide. Air is inhaled through the inhaling valve *T* and through the rubber mouthpiece *R*, which is firmly gripped by the teeth and lips of the wearer and is strapped securely to the head. Exhalation is through the tube and exhaling valve *S* into the breathing bag. The saliva trap *Z* catches the accumulated saliva. A small relief valve *K* enables excessive pressure within the bag to be reduced. This is accomplished by pressing the valve down with the finger. A pressure gage *P* enables the wearer to determine the amount of oxygen remaining. Each atmosphere of pressure indicates approximately a minute's supply of oxygen. Smoke goggles protect the eyes from irritating gases. The breathing bag and oxygen cylinders are attached to a harness that enables the whole apparatus to be comfortably carried. A by-pass valve *I* enables the wearer to secure oxygen in the event of stoppage of the reducing valve. A nose clip *Y* prevents the accidental breathing through the nose. Figure 317 illustrates the apparatus as worn. The weight is 32 lb.

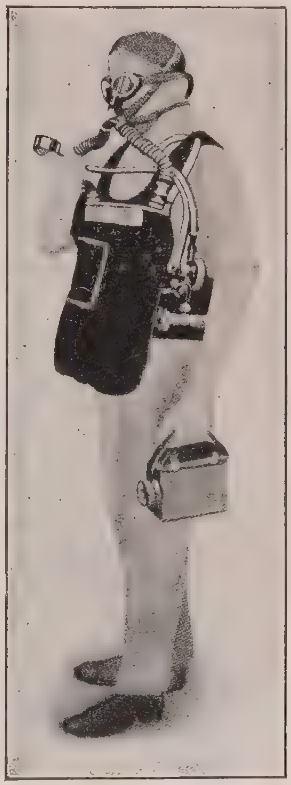

FIG. 317.—Proto apparatus in use.

The Draeger and Westfalia oxygen breathing devices have become obsolete in America. An apparatus similar to the Proto is sold to a limited extent. The Gibbs oxygen breathing apparatus, approved by the U.S. Bureau of Mines, is generally used in mining practice and has fulfilled the purpose as well as can be expected. It is shown in outline detail in Fig. 318. The capacity of the oxygen bottle is 270 liters at a pressure of 135 atmospheres. The oxygen supply is sufficient for a minimum time of 2 hr.; a maximum test, in which the wearer performed no work, records 10.5 hr. The weight is approximately 35 lb. In the older apparatus it was necessary to operate a by-pass when the exertions of the wearer required greater than the normal flow of oxygen, but in the Gibbs apparatus the flow of oxygen is automatic.

The oxygen flows through a closing valve into a reducing valve and thence past a safety valve through a brass tube in the cooler and then into the admission valve, through which it passes into another tube and into the exhalation side of the cooler, through the regenerator (containing caustic soda) and into the breathing bag and thence to the inhalation tube. When the breathing bag is full the admission valve closes, shutting off the flow and holding the oxygen in the reducing valve, where the

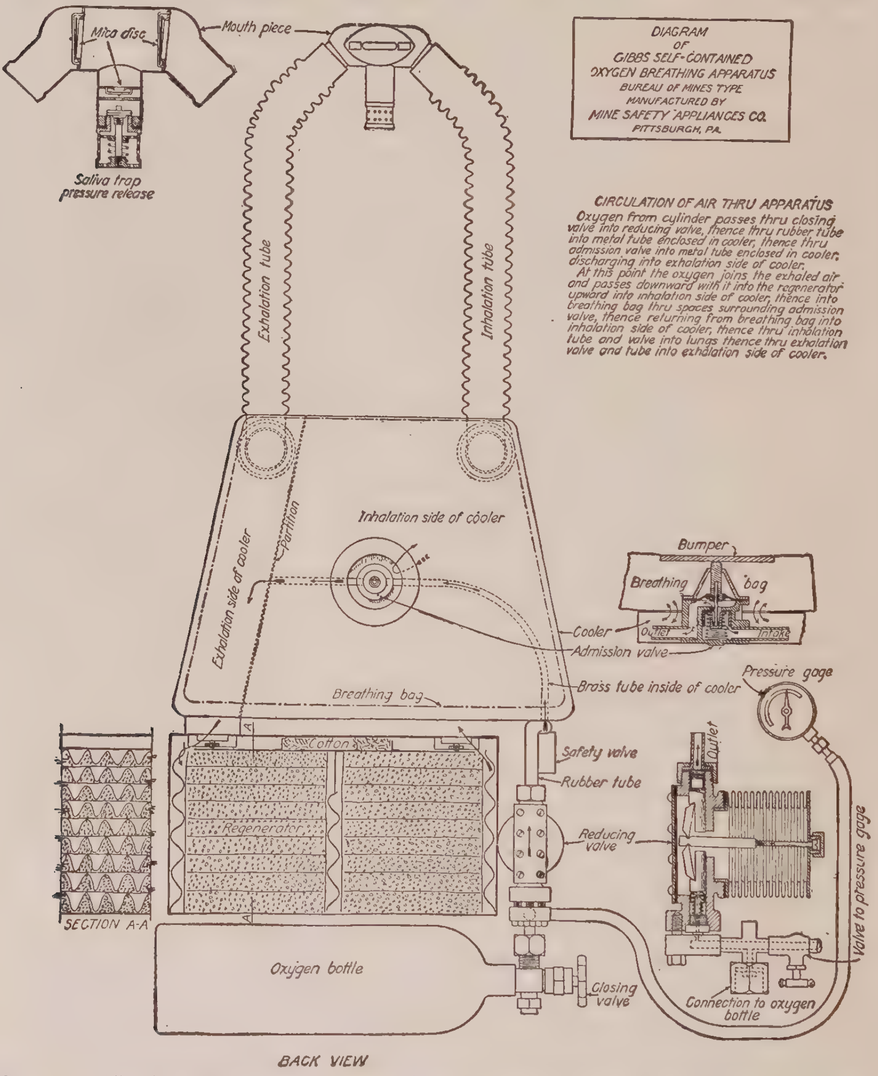
DIAGRAM
OF
GIBBS SELF-CONTAINED
OXYGEN BREATHING APPARATUS
BUREAU OF MINES TYPE
MANUFACTURED BY
MINE SAFETY APPLIANCES CO.
PITTSBURGH, PA.

CIRCULATION OF AIR THRU APPARATUS

Oxygen from cylinder passes thru closing valve into reducing valve, thence thru rubber tube into metal tube enclosed in cooler, thence thru admission valve into metal tube enclosed in cooler, discharging into exhalation side of cooler.

At this point the oxygen joins the exhaled air and passes downward with it into the regenerator upward into inhalation side of cooler, thence into breathing bag thru spaces surrounding admission valve, thence returning from breathing bag into inhalation side of cooler, thence thru inhalation tube and valve into lungs thence thru exhalation valve and tube into exhalation side of cooler.

FIG. 318.—Sectional view of Gibbs oxygen breathing apparatus. (*Mine Safety Appliance Co. of America.*)

pressure stretches the metallic bellows and shuts off the flow into the reducing valve. The oxygen gradually deflates the breathing bag and allows a bumper (weight) to rest upon and open the admission valve, replenishing the supply of oxygen. The mouthpiece contains two mica valves and a saliva trap, as shown in the figure. A nose clip and goggles are worn with the apparatus, which is carried by a harness strapped to the wearer. An outer case protects the entire apparatus. Figure 319 shows the apparatus as worn.

As accessories to the oxygen apparatus, an electric lamp of the storage-battery type and a life line are used. It is necessary to test out the apparatus thoroughly before it is used, in accordance with a standard procedure recommended by the manufacturers.

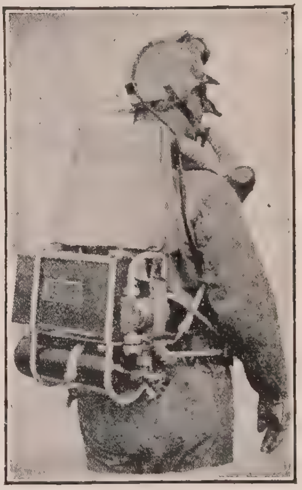

Fig. 319.—Gibbs oxygen breathing apparatus in position without cover. (*Mine Safety Appliance Co. of America.*)

Special training is required before breathing apparatus can be safely used. The usual practice is to select carefully a suitable squad and to require the members to use the apparatus in a special smoke chamber at regular intervals. It is customary for rescue men to practice regularly. Details of the training course required by the Bureau of Mines can be obtained from the Bureau.

The Bureau of Mines recommends that a rescue party should have not less than five members, preferably six. A relief station should be established at the last point where ventilation is effective, and here a relief crew equipped with breathing apparatus[1] should be stationed ready to go to the assistance of the first party. At a large mine at least four crews of six men each should be organized, two for outside and two for inside work. They should practice.

Breathing apparatus and smoke helmets are used in emergencies and must, therefore, be kept on hand and in proper working condition. They should be examined and tested at regular intervals. The rubber parts deteriorate rapidly and require close inspection. They should be a part of the equipment of all large coal and metal mines. Some state safety laws require a specific number of helmets in proportion to the number of miners.

For work in and about surface and underground fires where there is danger of carbon monoxide and where there is a sufficiency of oxygen, the Mine Safety Appliance Company has introduced the Burrell Mask

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[1] *T.P.* 82, *U.S. Bur. Mines*, describes apparatus.

No. CO-50. In this apparatus a mask is attached to the head of the wearer and connects with a corrugated tube, an exhalation valve, and a canister through which the air is drawn. In the canister the air is purified by passing through a chemical compound called Hopcalite, which is between two layers of moisture absorbents consisting of pumice, sodium hydrate, and lime. The carbon monoxide is converted to carbon dioxide and absorbed by the moisture absorbents, leaving the air in a state of comparative purity. The air then passes through a heat exchanger, which cools it, and thence into the tube leading to the mask. A timer indicates the exhaustion of the canister, which will give approximately 2 hr. continuous or intermittent use. Another canister can be quickly adjusted. The weight of the mask and attachments is 6 lb. An all-service mask is also made for use in ammonia fumes, industrial gases, carbon monoxide, and illuminating gas. Like the first described, it can be used only where there is a sufficiency of oxygen. In all other situations the oxygen breathing apparatus is the only safe apparatus, and the Bureau of Mines recommends this for mine fires and use after mine fires.

Accessory apparatus is used to some extent for the resuscitation of miners who have been rescued from a vitiated atmosphere or who have received severe electric shock. The lungmotor, pulmotor, Vivator, Pulvita, etc., are on the market. Respecting the use of apparatus of this kind, a medical committee selected by the Bureau of Mines disapproved of the use of mechanical devices for artificial respiration and approved of the use of manual methods. They recommended the use of a simple breathing bag connected to a supply of oxygen, provided with a mask fitting over the face and containing inlet and outlet connections and valves.[1]

PENSIONS, DAMAGES, WORKMEN'S COMPENSATION

Permanent disability or death as the result of a mine accident is settled by the mining company by giving a lump sum or pension or by damages awarded by a court of law to the worker or his dependents. Neither of these methods has proved to be wholly satisfactory, and workmen's compensation laws have been adopted by a number of states for the purpose of remedying the defects and securing equitable treatment to both employee and employer. Some of these laws have become cumbersome and exacting. State organizations administer the provisions of these laws, and some states maintain an insurance fund. Private insurance carriers often compete with state accident insurance and compensation organizations. Operators can usually elect to take out accident insurance coverage with either state or private companies. In all cases mine operators should be familiar with the specific provisions in the industrial compensation laws.

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[1] Report of the Committee on Resuscitation from Mine Gases. *T.P.* 7, *U.S. Bur. Mines.*

MINERS' DISEASES

Mine workers are subject to certain diseases, which will be briefly described.

Pneumonia is frequently the result of exposure to afterdamp produced by a mine explosion. It is more commonly the result of exposure to sudden changes of temperature. The best preventive is a convenient and comfortably warmed change house. Miners should be instructed to provide suitable overclothes so that when coming off shift and waiting at stations, which are usually cold and drafty, they can protect themselves. Change houses should be placed as close to shaft collars as possible.

Rheumatism is the result of exposure to heat, cold, and dampness. Care in respect to these matters serves as the best preventive. Miners often object to the use of water sprays where dusty drilling is done, saying that the moisture causes rheumatism. Whether or not this is so is doubtful.

Tuberculosis is associated with severe injuries such as crushed chest, strain, etc. It more frequently results from silicosis. In certain mining districts, especially in South Africa, it was a common disease. Tubercular miners should be removed from working crews, and working places, ladderways, and stations where miners congregate should be disinfected at intervals and kept clean.

Silicosis is produced by the continual inhalation of quartz dust. The lung tissue is weakened by the accumulation of fine, sharp particles of rock dust. The working efficiency of the miner is reduced and he is especially susceptible to tuberculosis. Wetting down broken ore where it has to be handled by shoveling, the use of sprays where upper holes have to be drilled, the use of hollow steel and water in drilling, the protection of the exhaust of rock drills, the practice of blasting at the end of the shift and giving sufficient time for the dust to settle, and the use of sprays at chutes will reduce the amount of floating dust to a minimum. In a dry, dusty mine these precautions should be taken. Good ventilation, good illumination, and short shifts are excellent preventives. Where dust cannot be prevented, respirators should be worn. Bag filtering units are sometimes installed at especially dusty zones and are effective.

Anthracosis or coal miner's lung is produced by coal dust. So far as is known, coal dust is not an irritant and its presence in the lungs produces no ill effects. All inhaled dust, however, is considered unhealthy.

Ancylostomiasis, or the hookworm disease, is apt to be found where the underground temperature is in excess of 78°F. and excessive humidity rules. It was prevalent in the tin mines of Cornwall and European coal mines. Its presence has been noted in California mines. The disease is parasitic in its nature. Cleanliness both of the person and of the underground workings, the sprinkling of salt solution about, and the avoidance of excessive moisture underground are said to assist materially in its control.

Lead poisoning, or painter's colic, sometimes attacks lead miners. Particles of lead mineral are accidentally taken with the food or get into the mouth from the dust that is inhaled. Thorough cleansing of the hands and person when coming off shift will to a considerable extent prevent this disease. The prevention of dust is essential.

Mercury poisoning results in most cases from the accidental taking of fine particles of the ore into the system through the mouth. Cleanliness of the person and the elimination of dust are the best preventives. Where native quicksilver occurs, lime-sulphur solution sprays may reduce the salivation hazard.

Typhoid is the result of unsanitary conditions. A well-protected water supply, whether underground or on the surface, is the first essential. The commissary should be protected from flies, and precautions should be taken to prevent the contamination of food. The sanitary arrangements on the surface and underground should be inspected at regular intervals. A liberal use of disinfectants should be encouraged. Carelessness in respect to these matters not infrequently characterizes small mining camps.

Malaria is a common disease, and its cause is well known. Protection from mosquitoes that carry the parasite is the first essential in the control of the disease. The next is the elimination of all breeding places of mosquitoes in the vicinity of living quarters. The first is done by screening doors and windows in living quarters, the second by drainage or by pouring crude oil on the surface of all pools of water. Malaria is especially prevalent in tropical countries and interferes with mining operations, often to a serious extent.

"Soroche" or mountain sickness is peculiar to high altitudes. The physiological effects of altitude are obscure. Some people are greatly affected, while others notice scarcely any effects whatever. The characteristic disturbances are headache, nausea, vomiting, and frequently nosebleed. They usually pass off in a short time, but where they persist the obvious remedy is to go to a lower altitude.

The engineering problem not only involves the steps required to control or prevent the diseases enumerated above, but also concerns itself with the preservation of the general health of the workers and the prevention of the diseases common in closely housed communities. Underground the sanitary precautions necessary are provision for dry closets on the main working levels in the proportion of one to every 15 or 30 workers. (The type selected should be such as to admit of convenient removal to the surface and ready cleaning. Closed buckets or cans are commonly used.) A supply of pure water for drinking purposes, drinking fonts of a sanitary type, and the removal of all lunch débris are necessary. At the surface change houses provided with locker, drying, and bathing facilities are essential. A sewerage system and facilities for garbage disposal are necessary adjuncts to the surface plant. Where mine camps or villages are required they should be carefully laid out. Camp sites should be selected on well-drained areas so located as to preclude the possibility of contaminating the water supply. Cesspools and a limited sewerage system should be provided. Garbage disposal should be effected by a furnace or deep pit at a considerable distance from the camp. Suitable garbage cans should be liberally used about the camp. Vaults and vault apartments should be tightly constructed and thoroughly screened and lime used liberally. They should be placed at a considerable distance from the camp.

Mess and cook tents or buildings should be thoroughly screened, and their entrances should be provided with short double-doored vestibules. Protection against flies and mosquitoes is required in all structures used as living quarters. Sanitary regulations should be posted at several points in the camp and systematic inspection inaugurated to secure their observance.

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CHAPTER XIX

EXAMINATION OF MINERAL DEPOSITS

Purpose.—The examination of a mining property is undertaken chiefly for the purpose of determining its value. The motive may be the purchase of the property, the making of an investment in the stock, the borrowing of a sum of money, or the taxation of the property. Examinations are also sometimes made for the purpose of settling the ownership of conflicting mineral rights, for the study of possible extensions of known ore bodies or the discovery of new ones, for the checking of operations of a going mine, or for the planning of new equipment and facilities. The general study of a mine from the standpoint of safe operation is another feature which has recently come into prominence and which offers a special field for the technical expert. Mining engineers, geologists, and civil, hydraulic, and electrical engineers make the examinations insofar as they apply to the special fields of each. The present chapter covers only the details of mine examination for purposes of valuation.

Types of Mineral Properties.—Mineral properties may be broadly classed into (*a*) undeveloped or prospective, (*b*) partially developed, and (*c*) fully developed properties. They may be further designated as gold, silver, gold and silver, copper, lead, lead and zinc, iron, and nonmetalliferous, such as coal, salt, and gypsum. Development workings may consist of test pits and boreholes or mine workings such as shafts, drifts, and crosscuts. The distinction first made implies that properties included under (*a*) afford only superficial information concerning the occurrence of the deposit, such features as size being unknown and accessibility for sampling being absent; that in properties under (*b*), access for measurement and sampling is possible in a part of the deposit only; that in (*c*) properties access for measurement and sampling is complete or almost so. Placer, iron, and disseminated copper deposits can in many cases be delineated and samples from them can be obtained by boring without the necessity of driving mine workings, and where it is possible to use this method the development and examination are simultaneous. Fissure deposits of gold, gold and silver, copper, and other ores usually cannot be satisfactorily delineated by borings, and development by mine workings should be initiated as a preliminary to valuation. From time to time as development proceeds the engineer can make his examination, or it may be carried out on the completion of a part or of all the development.

Preliminary Examination.—A thorough inquiry into the general features is a necessary preliminary to the examination. The extent of this inquiry will vary in different instances. At least the geology of the locality, the type of the deposit, the topographical conditions, the natural resources of the immediate neighborhood, the condition of near-by mines, and questions involving title should be reviewed. The result of the preliminary examination is to bring forth the obviously limiting conditions of the problem. The limiting conditions are physical, economic, and political. As examples of physical conditions may be mentioned access, absence of water, excessive quantities of water, no natural resources, scarcity of labor, climate, the small size of the deposit, scanty mineralization, structural features that would interfere with the mining operations, conflicting mineral rights, and difficulties in mining and ore treatment. Examples of limiting economic conditions are large initial capital expenditure, high operating costs, and small profits. Examples of limiting political conditions are unstable government, questionable protection of titles, antagonism of resident population, and high duties and taxes. By anticipating the general features in this manner the engineer is in a position to plan his methods more clearly and to avoid wasting his time in a detailed examination where the limiting conditions preclude the success of the enterprise at the very beginning.

Mine Sampling.—A reasonably accurate map is necessary as a preliminary to sampling. In a developed mine a map of the workings is usually available, and this is used as a base upon which to locate the position of the samples. In the absence of a map, an approximate one can be made with a Brunton pocket transit, or if greater accuracy is necessary a detailed survey is carried out and a map is prepared. Sampling need not be delayed, however, as known points can be selected to be tied in later by the survey, and from these points measurements can be taken to the sample cuts.

Where there is a definite structure, as in a vein, samples are taken by cutting grooves across the width of the vein at intervals of 5 to 20 ft. along the strike. The groove is so cut as to represent each part of the width in equal proportion. In a wide vein separate samples are made for each 4 or 5 ft. of width. This is done on account of the impracticability of cutting equal weights from each part of the width. In narrow veins the sample is taken across the full width. In practice, the cutting of the groove varies from a series of chips taken from an area from 4 to 6 in. wide to a definite shallow channel 4 or more inches wide and 1 in. or more deep. The nature of the vein filling and the conscientiousness of the engineer in charge determine the thoroughness of the work. Where the vein filling is soft and uniform it is a comparatively simple matter to cut a sample channel of more or less definite dimensions, but where it is hard and more or less fractured it is difficult to get a representative sample. In such a case, large pieces are knocked out; these must be broken and only enough of them to represent the place are placed in the sample. Alternating hard and soft bands present another difficulty. There is a tendency to get too much soft and not enough hard lodestuff in the sample. Where the banded structure is persistent, soft and hard bands can be separately sampled and measured. An erratic distribution of values requires large samples, *i.e.*, deeper and wider cuts than where there is a more uniform distribution. A massive vein or one in which any definitely marked mineral distribution is absent can be channeled without regard to the walls. Very small, rich veins of irregular width present an especially difficult problem. The sample width is sometimes taken from 3 to 3.5 ft. and necessarily includes waste from either or both walls. The alternative method is to cut the sample along the course of the vein, taking as the width of the cut the width of the vein. The procedure in any one case can be decided only by a study of the mineralization and by taking preliminary samples and panning or assaying them. The experience of the engineer is important in deciding on the best procedure.

The appliances used in cutting the channel are moils and hammers for hard rock and a small chisel-ended pick for a uniformly soft vein. The prospector's pointed pick is also used. Where compressed air is available a small air hammer such as is used by stonecutters is an excellent device. Under certain conditions the cuttings made by a rock drill can be taken for a sample. Two men are required, one to cut and the other to catch the sample in a box or bag. A light metal or wooden box is convenient for some positions; for others a canvas bag with the mouth distended by a wire hoop is used. A convenient arrangement is to narrow the bottom of the bag and leave it open. It can then be used as a funnel to fill the sample sack. When it is used to catch the sample the lower end is tied. Canvas spread on the floor of a working is also used.

The size of the sample taken will depend upon the dimensions of the cut. The weights per foot of cut are given in Table 131.

Table 131

	Dimensions of cut			
	½ × 3 in.	½ × 4	1 × 4	2 × 4
Wt., lb. per ft.	1.6	2.2	4.3	8.6
Wt., lb. per 5 ft.	8.0	11.0	21.5	43.0

It is difficult to lay down any special rules respecting the weight of sample taken from each cut. Some engineers limit the weight to 10 lb., others to 20, while still others take a series of chips along the sample length without regard to any limit weight. Interrelated with the size of the sample is the spacing between sample cuts, concerning which two opposing opinions may be mentioned. One favors the taking of many closely spaced samples, each of small weight, while the other favors large samples spaced at longer intervals. On the whole, it is somewhat easier to take numerous small samples than large ones, and in many cases the theory of probability favors the former opinion. Probably the most satisfactory method of deciding the question is to take several trial samples from a representative part of the vein and compare the results of assays.

The sequence of operations in taking samples is as follows.

1. Mark the sample intervals along the drift by chalk or paint.
2. Clean the strip at the place where the sample is to be cut, using water and a coarse brush.
3. Cut the sample; crush and reduce by passing through a Jones sampler if necessary.
4. Place the sample portion in a numbered canvas sack.
5. Record the number and position of the sample.
6. Measure the width of the vein at the sample cut; determine the dip.
7. Record any peculiarities of mineralization, or other points that might be pertinent.
8. Make sketches of the vein structure at places where there are significant changes from normal conditions.

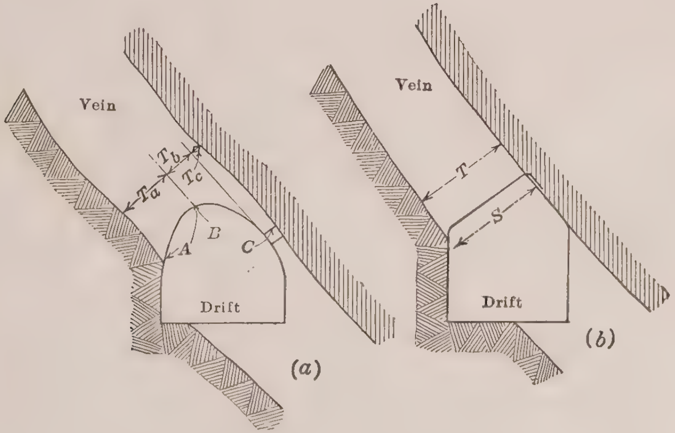

Fig. 320.—Sampling irregular surfaces.

The measurement of the thickness of the vein and the taking of the sample is sometimes complicated by the conditions shown in Fig. 320. In *a* the exposure of the vein in the back part of the drift would necessitate the taking of three separate samples A, B, and C. The thickness of the vein represented by each sample would be measured and the value across the full width of the vein would be determined by calculation. The alternative would be to square the exposure of the vein from wall to wall, as shown in *b*, before making the cut. Check samples are taken at regular intervals. These consist of taking a second sample from the groove or cutting another alongside. How many check samples should be taken is a matter of judgment. In an important examination more would be required than under other conditions. Check samples should be taken after systematic sampling and by a different sampling crew. Samples of waste are also occasionally taken and placed along with the regular samples, in order to detect any attempt at systematic salting. Occasionally a second set of check samples is taken with the same purpose in view.

Samples for mill runs range from 10 to 50 tons. They are taken by drilling and blasting portions of the vein. The sample may be taken from a number of points evenly spaced along a given ore shoot, or it may be broken down more or less uniformly from the whole exposure of the ore shoot along a given level. It serves as a check upon the smaller samples and of course will be taken only where milling facilities are available.

The sampling of gold deposits where the gold is erratically distributed presents peculiar difficulties. S. J. Jennings describes an example of a mine in Alaska in which the ore was supposed to carry $2 per ton in gold value. Successive channel samples at 5-ft. intervals ranged from zero up to $12 per ton. Samples were finally cut over a strip 2.5 ft. wide along a crosscut for the full height. Each sample from a 12-ft. crosscut weighed 30 tons. The samples were crushed in a 5-stamp battery provided with a slotted screen equivalent to 60 mesh. The pulp was concentrated on a Wifley table, the tailing from the table being carefully sampled. The concentrates were amalgamated in a barrel, and the tailing from the barrel was sampled. The sample value was computed from the weight of gold obtained, the tailing value of the table, the tailing value from the barrel, and the weight of the respective products. Results were obtained which checked. W. W. Mein, on the other hand, found that a large number of channel samples, taken from cuts 4 by 3/4 in. spaced 5 ft. apart, gave reliable results at the Dome mine (Canada).[1]

Samples from boreholes are in the form of fine cuttings or sludge and cores. Where the churn or jetting drill is used, the sample is in the form of sludge; with the diamond or chilled-shot drill, part of the sample may be obtained as a core and part as sludge or the entire sample may be obtained as a core. As the bore penetrates the ore body a sample is taken for every 5 ft. of depth. In boring placers, the samples are taken for each foot of depth. The spacing of boreholes is determined by the type of deposit and the judgment of the engineer. Mining practice has established more or less well-defined limitations. In boring disseminated copper ore deposits, for example, boreholes are put down at the intersections of a coordinate system, the unit of which is 200 ft. In the iron-ore deposits of the Mesabi Range, after a discovery has been made upon a 40-acre tract, bores are sunk at 200- or 300-ft. intersections of a coordinate system, the unit of which is 100 ft. In the Oroville district (California), dredging ground is divided into 5- to 10-acre blocks and a single hole is drilled in the center of each block. In other instances the surface is divided into squares from 200 to 400 ft. on each side and a single hole is drilled in the center of each square. At Breckenridge, Colorado, holes were sunk 250 ft. apart. In Alaska, placer deposits in which the pay streak was erratically distributed required a shaft to each acre.

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[1] *Bull.* 78, *Min. Met. Soc. America*, p. 177.

In copper, lead, and zinc deposits, sludge samples obtained by the use of the churn drill are generally accurate. Their accuracy depends upon the absence of any concentration caused by either the elimination of light gangue material or the breaking down of rich masses of ore from the sides of the bore. If the borehole is dry, only sufficient water is used to make the sludge fluid. If enrichment takes place from the sides of the bore, casing must be used and kept driven well down toward the bottom. If the bore is filled with water to any considerable depth, the churning action of the drill may serve to suspend more or less of the lighter particles of sludge and cause a moderate enrichment of the sample. This source of error may be expected to be more or less compensated for as the bore is deepened. In testing the accuracy of bore sampling in the Ajo district (Arizona), test pits and raises in sulphide copper ore gave results within an average of 0.05 per cent of the samples obtained by diamond drilling. Drift samples averaged 0.26 per cent higher than the assay value of blocks of ore, as indicated by drill holes at the corners of the block. It is of interest to note that samples on the same working, obtained from test pits by taking every tenth bucket windlassed out, averaged 0.15 per cent higher than channel samples cut on the completion of the pit.[1] A raise extended upon a churn-drill hole on the Sacramento Hill property of the Copper Queen Mining Company (Arizona) gives the following comparison.

Churn-drill Sample, Per Cent Copper	Channel Sample from Raise, Per Cent Copper
3.66	4.00
3.72	2.50
4.18	3.40
2.38	5.60
5.78	4.50
4.25	3.70
2.82	2.80
3.30	2.90
Average 3.80	3.79

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[1] *Trans. A.I.M.E.*, vol. 49, p. 605.

The channel samples were taken from 5-ft. lengths and the weight of the sample approximated 8 lb. per ft. of channel.[1] The comparison indicates differences of considerable magnitude, but these differences are almost compensated for when the average is taken.

In sampling iron-ore deposits where the jetting drill is used, as is the practice in soft iron-ore deposits, the stream of water and cuttings are caught in several barrels. The coarser sludge settles rapidly; and unless steps are taken to settle the finer particles in suspension, the sample is liable to enrichment or impoverishment, depending upon the nature of the particles in suspension. Some engineers make allowance for the resulting error by comparing the results of a borehole and samples taken from a raise. The ratio of the average of both sets of samples is taken as a factor to apply to all other drill-hole averages upon the property. Where it is impracticable to handle all the water and sludge, at least several tests should be made of the accuracy of samples taken with a moderate degree of settling, compared with samples that have been completely settled. The careful sampler takes nothing for granted, but tests the accuracy of his samples experimentally. Nevertheless, where methods in a given district have been shown to be reasonably accurate, the practice thus established can be followed.

The sampling of placer deposits by churn drills has its peculiar difficulties. There is always the liability of enrichment or impoverishment from the sides of the bore. This is largely obviated by using casing and by drilling only a foot in advance, driving the casing, and then removing the sludge with a suction bailer. Where the gravel permits, the casing is driven in advance of the bit. The results of drilling are given in cents per cubic yard, and the volume of gravel removed is estimated by the displacement of the casing shoe for each advance. It is obvious that in most cases slightly more gravel will be removed than is indicated by the displacement of the shoe. For example, the displacement of a cutting shoe 7.5 in. in outside diameter is approximately 0.01 cu. yd. per foot of advance (more exactly, 0.0113). The use of this volume would give results slightly higher than actual. By comparing the results of boreholes with test pits, Radford showed that the displacement of a 6.5-in. diameter casing should be figured as 0.27 cu. ft. instead of 0.23, the calculated displacement. This would give an increase of about 17 per cent of the calculated displacement. While this percentage increase could be used in the absence of more specific information, the engineer should test its application by comparing the results obtained with results from one or more test pits. To guard against the removal of excessive quantities of gravel, the amount removed for each advance should be approximately measured before it is washed.

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[1] A. Notman, *Trans, A.I.M.E.*, vol 52, p. 457.

In diamond drilling in ore where complete cores are obtained, the core is used as the sample. Where only part of the core is obtained the sludge must also be saved. This can be done by filtering the flow from the drill hole through heavy burlap sacks. In working down the sample for a given length of bore, both sludge and core can be combined or both can be sampled and analyzed separately. Where separate samples are made, the unequal division of sludge and core necessitates the calculation of results. This is done by multiplying the respective results by the volumes of sludge and core, adding and dividing by the combined volume. To facilitate the computation of volumes, a diagram similar to that shown in Fig. 321 can be prepared and used.

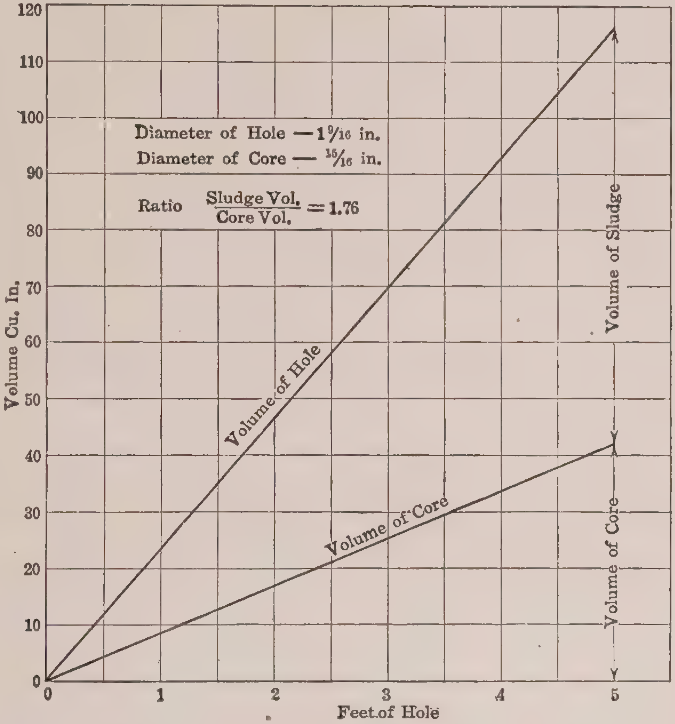

FIG. 321.—Diagram used for determining proportions of sludge and core.

Grab samples are taken by picking a number of pieces of ore from a pile, the pieces being taken from points uniformly spaced over the pile. They are of service only where an approximate estimate of value is required or where there is a question as to the advisability of incurring expense for accurate sampling.

Shovel samples are taken by reserving every fifth, tenth, or twentieth shovelful for the sample. This method of sampling can be used where test pits are sunk. It gives usually an accurate sample. A similar method is the reserving of every fifth or tenth bucket of ore hoisted from a test pit.

Specimen samples consist of picked lumps of ore that show marked characteristics of the ore body. They are selected to represent either the average grade of ore or some peculiar feature. They are of no value for the quantitative determination of value but are of considerable use in making studies of mineralization and for determinations of specific gravity. They also serve for the preliminary studies of the ore treatment.

Sampling Placer Deposits.—Where the gravel bank is exposed from bedrock to the top of the bank, sample cuts can be taken at intervals along the exposure. The cut is carefully made, a wooden template being used to secure accurate dimensions. A canvas sheet is spread at the bottom to receive the cuttings, and a second piece of canvas is sometimes used to prevent the loss of flying particles. The coarse material is separated by hand and the fine measured by the number of miner's pans. All the fine material is panned and the gold is recovered by amalgamation. A rocker is more convenient than the pan. The cut can be sampled a foot at a time, or it may be finished, all the material being worked down in one batch.

Where the bank is not exposed, test pits are used where water is absent and churn-drill holes where it is present. The test pit is 3.5 ft. in diameter, and all the material removed is used as a sample. A rocker or small sluice is necessary because of the bulkiness of the material. A wooden template is used to secure an accurate measure of volume. Channels are sometimes cut in the wall of the shaft. Vertical distribution of the valves is determined by washing the excavated material in batches, each of which represents a foot or more of depth. Shafts and boreholes are placed in lines at right angles to the direction of stream flow. They are more closely spaced in this direction than in the direction of stream flow. Test pits are as a rule more satisfactory than boreholes, since a shaft 3.5 ft. in diameter will be equivalent to 30 or more drill holes in the volume of gravel afforded for washing.

Where values are concentrated on bedrock, crosscuts are driven along the bedrock at right angles to the stream flow. These afford information as to the width of the pay lead. As the values in many cases penetrate the bedrock a foot or more, it is necessary to remove the upper surface of the bedrock as well as the lower stratum of gravel. In drift mines the returns are sometimes given in terms of value per square yard of bedrock.

The washing of gravel samples is effected by pan, rocker, or sluice. Each sample is represented by a separate cleanup. The value of the separate cleanups from a given shaft or bore is estimated by counting the colors. To check this estimate the individual portions are combined and amalgamated, and the amalgam is dissolved in nitric acid. The resulting gold is washed, ignited, weighed, alloyed, parted, and reweighed. The fineness of gold varies in different placer deposits, so that its determination on one or more portions is essential.

The compactness of the gravel, the proportion of fine material, the prevalence of large boulders, and the presence of clay are noted at each point where a pit or bore is sunk.

Sampling Copper Deposits.—Where development workings are accessible, the channel method of sampling is employed. Where samples are obtained from churn or diamond-drill holes, the sludge from the former is reduced to about 50 lb. weight (dry) by running it through a sample cutter of the Jones type. The sample portion is received in a galvanized-iron tub. It is then dried, sacked, and sent to the assay office. The determinations usually made are copper, iron, silica, and sulphur. Occasional complete analyses are also made. The sludge is panned and the copper minerals present in each sample are determined. A detailed log of each bore is kept.

Sampling Iron-ore Deposits.—The channel method of sampling is used in accessible workings. In soft ore bodies the jetting drill and in hard ore bodies the diamond drill are the usual methods for sampling from the surface of a deposit. The sample from the jetting drill is settled, surplus water is decanted, and the residue is dried in a metal tub or pan. Then it is cut down and a smaller sample is taken, or else the entire sample is sacked and sent to the chemist. Iron and phosphorus are determined on each sample. In some cases silica and alumina are also determined, and complete analyses of some of the samples are occasionally made. Where manganese occurs, its determination is essential. The hardness and texture of the iron ore are made a matter of record where the conditions permit of the determination of these features.

Sampling Coal Seams.—In accessible workings samples are taken at widely spaced points and the thickness of the seam is measured. The layers of coal and foreign material are also measured. Where it is practicable to separate a parting in mining operations, the coal benches are separately sampled by channeling. Where this is not practicable, the channel is cut from roof to floor. The coal sample is reduced in the usual manner, and the ash, sulphur, volatile combustible, and coke are determined. Calorimeter tests of heating effect are also made. Large samples are taken for washing tests. Small samples can be used for "float and sink" tests that indicate the amount of removable foreign material. The hardness and texture and the presence of fracture planes are noted. Weathering tests of the coal are essential in some instances.

The directions for taking coal samples in use by the U.S. Geological Survey are given in the following extract.[1]

> 1. Select a fresh face of unweathered coal at the point where the sample is to be obtained and clean it of all powder stains and other impurities.
>
> 2. Spread a piece of oilcloth or rubber cloth on the floor so as to catch the particles of coal as they are cut and to keep out impurities and excessive moisture where the floor is wet. Such a cloth should be about 1½ by 2 yd. in size and should be so spread as to catch all the material composing the sample.
>
> 3. Cut a channel perpendicularly across the face of the coal bed from roof to floor . . . of such size as to yield at least 6 lb. of coal per foot of thickness of coal bed; that is, 6 lb. for a bed 1 ft. thick, 12 lb. for a bed 2 ft. thick, 24 lb. for a bed 4 ft. thick, etc.
>
> 4. All material encountered in such a cut should be included in the sample, except partings or binders more than ⅜ in. in thickness and lenses or concretions of "sulphur" or other impurities greater than 2 in. in maximum diameter and ½ in. in thickness.
>
> 5. If the sample is wet, it should be taken out of the mine and dried until all sensible moisture has been driven off.
>
> 6. If the coal is not visibly moist, it should be pulverized and quartered down inside the mine to avoid changes in moisture, which take place rapidly when fine coal is exposed to different atmospheric conditions. The coal should be pulverized until it will pass through a sieve with ½-in. mesh, and then, after thorough mixing, it should be divided into quarters and opposite quarters rejected. The operation of mixing and quartering should be repeated until a sample of the desired size is obtained. When the work has been properly done a quart sample is sufficient to send for chemical analysis. This sample should be sealed in either a glass jar or a screw-top can with adhesive tape over the joint and sent to the chemical laboratory for analysis.

Sampling of Tailing and Waste Dumps.—Tailing dumps are preferably sampled by augers. The material from a given bore is thoroughly mixed and quartered down to a sample of from 10 to 50 lb. Boreholes are spaced at from 25- to 50-ft. centers. The depth of the bore or the thickness of the pile is measured at each point. Assays or analyses are made. Waste dumps or low-grade ore dumps are sampled by either pits or crosscuts. Every tenth shovel is taken for the sample portion.

Salting.—Artificial enrichment of ore faces or samples is called "salting." Its purpose is obvious, and engineers have to be very much on their guard to prevent tampering with their samples or assays. Ore faces must not only be carefully cleaned at the sampling cuts but a careful examination of the ore itself must also be made. It is essential to have only trustworthy men on the sampling crews. Sample bags are gathered up and taken along with the sampling crew until a sufficient number have been collected to warrant a trip to the surface, where the samples are turned over to the sampler for reduction or are stored in a safe place. Assay reagents should be tested where assays are made at the mine. The judicious intermixture of barren samples or samples containing known values with the mine samples will usually detect salting. Care and watchfulness are the principal safeguards. The sealing of sample sacks and storerooms where samples are temporarily placed is a precaution often taken. Salting is usually limited to gold and silver deposits. Other metalliferous deposits are sometimes salted but on the whole present a more difficult chemical problem to the dishonest.

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[1] *Bull.* 537, *U.S. Geol. Survey*, p. 106.

Computation of Averages.—The calculation of average values from a series of samples is best illustrated by assuming an example. Let the following weights and values of four different lots of ore be given:

(*A*) 100 tons averaging \$10 per ton
(*B*) 50 tons averaging \$ 5 per ton
(*C*) 75 tons averaging \$12 per ton
(*D*) 90 tons averaging \$ 8 per ton

The average value of the four lots combined is equal to the sum of the products of weight and value per ton divided by the combined weight. The computed average value is \$9.11 per ton. Weight may be considered as the product of length, width, thickness, and density factors, pounds per cubic foot. Four separate equations may then be written.

$$
\begin{aligned}
(A)\quad \text{Total value} &= L_a \times W_a \times T_a \times Q_a \times 0.0005 \times \$10 \\
(B)\quad \text{Total value} &= L_b \times W_b \times T_b \times Q_b \times 0.0005 \times \$5 \\
(C)\quad \text{Total value} &= L_c \times W_c \times T_c \times Q_c \times 0.0005 \times \$12 \\
(D)\quad \text{Total value} &= L_d \times W_d \times T_d \times Q_d \times 0.0005 \times \$8 \\
\text{Combined value} &= A + B + C + D \\
\text{Average value} &= \frac{\text{combined value}}{\text{summation } L \times W \times T \times Q\ 0.0005} \\
0.0005 &= \frac{1}{2{,}000} \text{ or reduction of pounds to tons} \\
Q &= \text{pounds per cubic feet}
\end{aligned}
$$

It is evident that 0.0005 is a common factor in both numerator and denominator and hence may be eliminated. If the ore is of uniform density the factor Q becomes a common factor and is eliminated. If the spacing between samples is uniform the factor W can be eliminated. If the spacing between levels (involving the factor L) is uniform, each sample would then represent half the distance between levels; this would be a common dimension for all and could be eliminated. If the thickness of the vein were constant this dimension could be eliminated. In the event of all dimensions of the four lots being the same, the arithmetical average of the separate values would be the average value of the lot.

The conditions for the four cases are graphically represented in Fig. 322. In most cases of sampling veins the sample intervals are the same, the length of each separate block is likewise constant, and the density of the ore is sufficiently close to a constant value to be disregarded. The thickness of the vein and the value are the two variables that must be taken into account. The average is obtained by adding the products of the respective widths and values and dividing by the sum of the widths. If the sample intervals are irregular, then each sample cut is assumed to

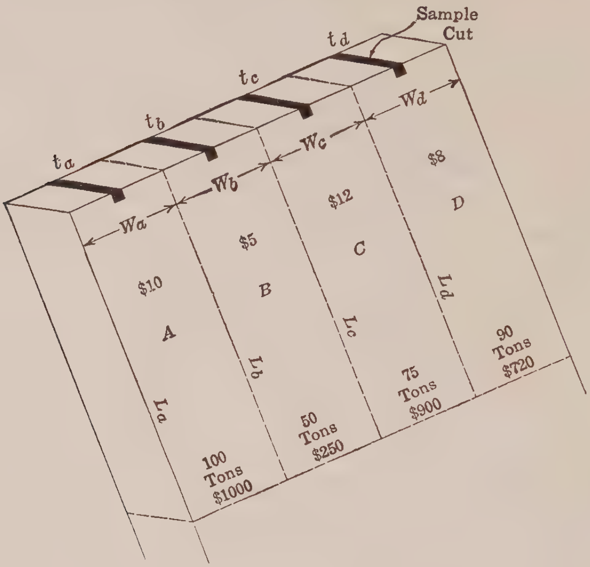


Fig. 322.—Graphical representation of details of a series of samples.

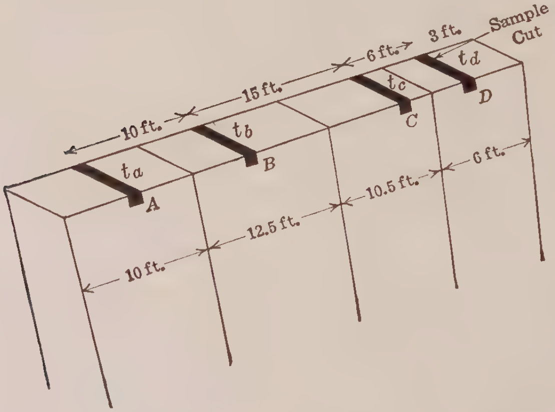


Fig. 323.—Unequal spacing of sample cuts.

represent the vein on either side up to a point midway between contiguous sample cuts. Figure 323 illustrates the case. The average is the sum of the product of the thickness, width, and value for each cut divided by the sum of the products of width and thickness for each cut. In Fig. 324 the average of a number of samples taken between and including the two end samples is figured by taking the sum of the products of widths and values for the sample cuts *B* to *H* inclusive and half the sum of the products of width and value of the end cuts *A* and *I*. This sum, divided by the sum of the thickness of sample cuts *B* to *H* inclusive plus half the sum of the thickness of sample cuts *A* and *I*, will represent the average value. It is customary to disregard the halving of the end samples where there are more than a nominal number of samples.

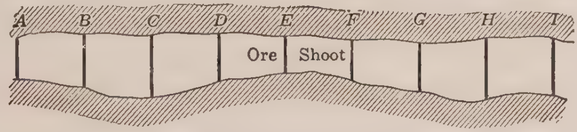
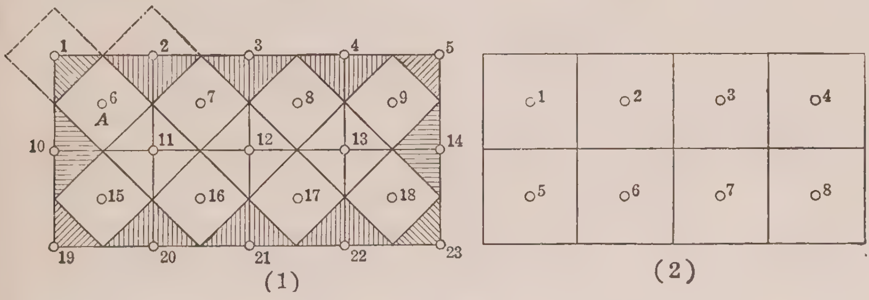


Fig. 324.—Equal spacing of sample cuts.



(1) (2)

Fig. 325.—Sampling by boring.

The average value for a single borehole is figured by taking the arithmetical average of the separate samples where each represents a constant sampling length or, where this length varies, by multiplying the respective sample values by the lengths represented by each sample, adding and dividing by the total length sampled. The averaging of a series of boreholes admits of several variations. If the boreholes are at the intersections of a system of coordinates, we can assume that each square will be represented by the four holes on the corners. The average for the square is obtained by the sum of the products of the average value of each hole and the sample length divided by the sum of the sample lengths. The average for a series of contiguous blocks would be the sum of the products of each block's average value and average depth divided by the sum of the average thickness (depth) of the blocks.

In Fig. 325, (1) is an illustration of a somewhat different method. Blocks 6, 7, 8, 9, 11, 12, 13, 15, 16, 17, and 18 are given a weight of 1; blocks 1, 5, 19, and 23 a weight of ¼, and blocks 2, 3, 4, 10, 14, 20, 21, and 22 a weight of ½ in computing the average for the area represented by the figure. In (2), each block is represented by a single borehole in the center. The computation of the average value in this case follows the method first described. In the three cases the first would require 15 boreholes, the second 23, and the third 8 for the area shown in the figure. Of the methods shown in Fig. 325, (1) would give the greater degree of accuracy.

If the sample, whether from a channel or borehole, is regarded as representing a volume or weight as illustrated in the first paragraph of this division, little difficulty will be experienced in calculating averages. The principle that each borehole includes an area bounded by lines drawn through the midpoints and at right angles to the lines connecting contiguous boreholes will enable the areal extent represented by each borehole to be rapidly determined. The same principle modified as described before applies to channel samples taken across veins.

The five possible cases that arise in determining averages are summarized in the following rules.

1. Weight × value of sample or average of bore
2. Volume × value of sample or average of bore
3. Area × value of sample or average of bore
4. Thickness × value of sample or average of bore
1. Divide sum of products by sum of weights.
2. Divide sum of products by sum of volumes.
3. Divide sum of products by sum of areas.
4. Divide sum of products by sum of thicknesses.
5. For arithmetical average divide sum of sample values by number of samples.

In arriving at averages certain samples are sometimes abnormally high as compared with contiguous samples, especially in gold and gold and silver deposits. Engineers differ in their methods of computing averages under such circumstances. Some reject the high result and use an average of the two neighboring samples. Others resample, and if the same result is obtained use it in their computation.

Volume.—The volume of ore developed in a vein is computed by measuring, by means of a planimeter, the projected area of the ore body on the map, multiplying by the secant of the average dip angle and by the average thickness of the ore body. In Fig. 326 the plan of two ore shoots is illustrated. The limits of the ore shoot between levels are obtained by connecting with straight lines the outer limits of the ore shoot on each level with the outer limits on the levels above and below. It is seldom that an ore shoot will fall exactly between such lines, as ore shoots are as a rule irregular in outline, but these limits are assumed in the absence of more specific information. Separate determinations of the ore between neighboring levels can be computed in the same manner. The continuity of an ore body between levels is more or less a matter of uncertainty unless the ore body is divided at points from 100 to 200 ft. apart by raises. Even in cases of this kind, horses and other irregularities may occur and render the computation inaccurate.

It is customary to make certain distinctions, in the different blocks of ore, on the basis of continuity. Ordinarily a block of ore contained between two levels 100 ft. vertically apart and exposed on the ends by raises is considered to have the continuity within the four exposures proved or reasonably certain. The continuity is less certain where the ore block is exposed on three sides, and still less so where exposed on two sides. Where it is exposed on one side only, continuity beyond a reasonable footage is very uncertain. Continuity cannot be considered as an abstract factor but takes into consideration the strength of the vein, the persistency of mineralization, the type of deposit, structural features such as faults and folds, and the presence of limiting formations such as an impervious stratum or a later intrusive. It is one of the troublesome factors in mine examination and requires experience and a broad knowledge of the geological characteristics of different ore bodies. In Fig. 326 the different ore blocks with the exception of *A*, *B*, and *C* are exposed on two sides. If the ore body is strongly mineralized and the walls are well developed, the ore blocks can be assumed to be proved; but if the walls are uncertain and the mineralization is erratic, the ore will be classed as "probable ore." In a strong ore body the extensions of the

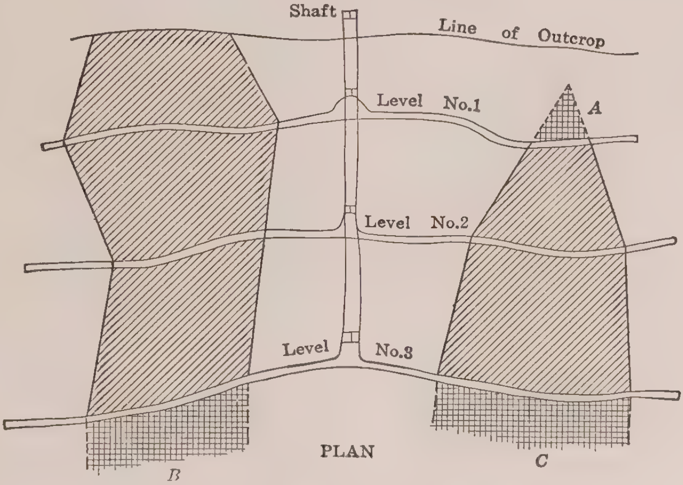


FIG. 326.—Classification of ore blocks.

ore shoots, blocks *A*, *B*, and *C*, might be considered as proved ore to a distance of from 50 to 100 ft. below the lowest level. Under opposite conditions the ore in these blocks would be classed as probable or "prospective ore," and as such might be allowed an extension beyond the lowest level of from 25 to 50 ft. "Proved," "probable," and "prospective" are terms of degree and measure the probability of continuity. They are sometimes exactly defined as ore blocks exposed on four sides, ore blocks exposed on two and three sides, and ore blocks exposed on only one side. While such definitions may answer in some cases, they cannot always be applied, since an ore block exposed on only two sides can be classed as proved ore under favorable geological conditions, while in other cases ore cut on four sides would justly be termed probable ore.

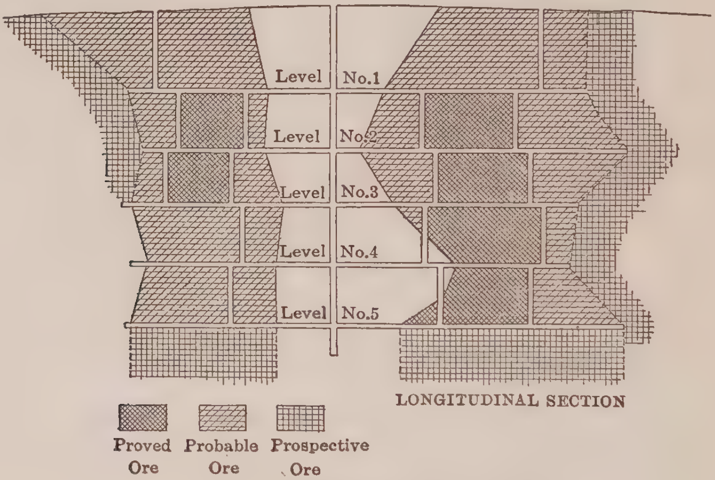


Fig. 327.—Classification of ore blocks.

Figure 327 illustrates the different types of ore blocks and illustrates the distinction between proved, probable, and prospective ore. The much-abused term "ore in sight" is frequently used and may signify only proved ore or proved ore and probable ore together. Exact definition of the terms used in ore classification is essential in a mining report.

The volume of irregular masses of ore can be computed approximately if sufficient measurements are available to determine the average length, width, and thickness. The product of the three average dimensions will give the approximate volume. Where the dimensional features of a massive or lenticular ore body are determined from borehole data, the computation of volume is based on the construction of accurate sections of the ore body spaced at equal distances transversely to the main axis. The procedure is to measure the area of each section with a planimeter. The volume of ore between each pair of sections is the product of the mean area of the sections and the perpendicular distance between the vertical sections. The total volume is the sum of the separate volumes. Where a sufficient number of bores have been placed, an accurate estimate of volume is easily made.

Figure 328 represents a hypothetical case in which the outer limits of the ore body are unknown but in which sufficient bore hole data for a volume estimate are available within the limits of the rectangle shown on the plan. The transverse sections are shown in the figure. The volume within the rectangle is calculated by the mean-area method described before. The extension of the ore body outside the limits of the rectangle is a matter of conjecture. Some engineers would allow a lateral extension equal to the thickness of the ore shown in the outer drill holes and a longitudinal extension on either end equal to one-half the distance between the sections. The additional ore volume would be classed as prospective ore. As in the case of the ore shoot in a vein, lateral and longitudinal extension would be determined largely by a study of geological conditions and the habit of similar deposits occurring under like conditions.

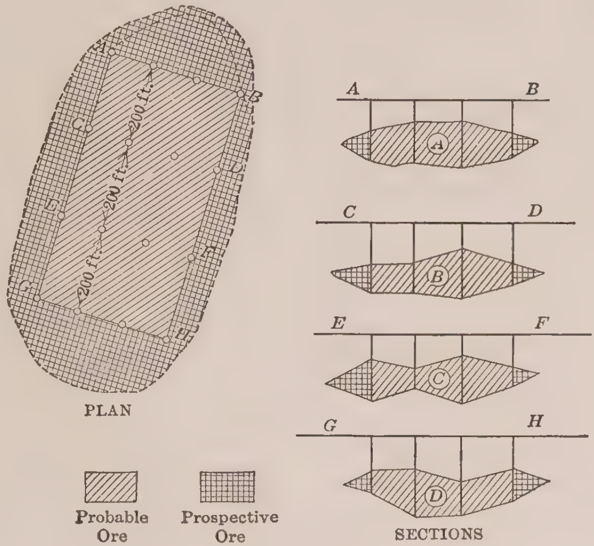


Fig. 328.—Extensions of known ore bodies.

Density and Weight.—The reduction of the volume to weight requires the density of the ore to be known or experimentally determined. This is readily done in the case of compact ores by making a number of specific-gravity determinations on representative specimens of the ore. Porous ores present a problem. Where a known volume of ore can be cut out, weighed, dried, and reweighed, the density of the ore in place can be determined quite accurately. Where it is impracticable to cut a known volume, approximate determinations can be made upon representative lumps by the sand-displacement method. The sand used must be thoroughly dry and fine. After the volume is determined, the lump should be broken up, thoroughly dried, and reweighed. From the volume and dry weight the density can be computed.

All samples for analysis and assay are dried at from 100°C. to 110°C. for at least 1 hr. The weight of most metalliferous ores is expressed by the short ton of 2,000 lb.; that of coal and iron ore in long tons of 2,240 lb. Where the metric system is used, the metric ton, 1,000 kg., equivalent to 2,204.6 lb., is the unit. Where doubt exists as to the general applicability of the density factor, a reasonable factor of safety should be applied to the tonnage estimate.

Gross Value.—The gross value is the product of weight and average value. The average value is computed from the market price of the metal or mineral. Except with gold, metal prices are subject to wide variation, and it is evident that the gross value will show corresponding variations. Average metal prices extending over the last 10-year period are sometimes used. Some engineers use the average metal price for the year immediately preceding the valuation. Abnormal metal prices should not be used, and the engineer should study the trend of prices of the particular metal and select a conservative figure.

Recoverable Value.—It is seldom that all the ore in a given ore body is recovered in mining. The percentages have been given under Mining Methods. Usually no allowance is made for this factor, as the quantity estimate may be so conservative as to provide ample allowance for 5 or 10 per cent loss in mining. An allowance should be made where the mining loss exceeds these figures.

In the treatment of an ore by ore-dressing methods, the cyanide process, or other methods, the percentage recovery ranges from 60 to 95 per cent. Just what the percentage will be in any given case can be determined only by experimental tests and mill runs. In examination work the problem of ore treatment is carefully considered, since it is one of the limiting conditions. Where similar ores are being treated the percentages obtained can be assumed to apply to the particular ore. Tests should be made where there is any doubt. Such tests should be so conducted as to give the percentage recovery under practical working conditions.

Sale of Product.—The product of a mine may be bullion, metal, concentrates, mineral, or ore. All are sold either directly to consumers or through agencies. Certain commercial requirements must be met, and the terms of payment and the penalties exacted for variation from commercial requirements or for various impurities differ for different metals and products. In the introductory chapter examples of the conditions under which mineral products are sold were given and need not be repeated here. The difference between the gross recoverable value and the sales value is sufficiently large to require thorough investigation of the conditions under which the product must be sold. Freight and sampling charges, agent's commissions, and all other expense involved in the marketing of the end product when deducted from the gross sales value will give the net sales value.[1]

Net Value.—The net value of the unit of ore is the difference between the net sales value and the cost of mining and ore treatment. Net value as thus defined is the profit obtainable from a unit of ore. With very few exceptions, either a mine plant or mine and ore-treatment plants are necessary before ore can be produced or the products therefrom placed upon the market. An initial investment of capital is required to provide the facilities for working. The capital requirements are influenced by the size of the plant, the locality in which it is erected, and the cost of machinery, supplies, and labor. The cost of operating the plant is influenced by the methods used and the cost of labor, supplies, and power. The following analysis indicates the more important features of the cost of the plant and the operating costs.

I. Plant Cost—Mine Plant, Ore-treatment Plant
  First cost:
    1. Preliminary engineering, preparation of plans and specifications
    2. Machinery, supplies, and building material
    3. Freight and transportation
    4. Erection
  Upkeep cost:
    1. Repair and maintenance
    2. Improvements and extensions
  Overhead cost:
    1. Interest and depreciation on plant investment
    2. Insurance and taxes
II. Operating Cost—Mine and Ore-Treatment Plants
  Mine:
    1. Labor
    2. Supplies } Breaking, support, transportation, development, illumination, ventilation, drainage
    3. Power
  Treatment plant:
    1. Labor
    2. Supplies } Crushing, sizing, concentration, ore treatment
    3. Power
III. Superintendence and Control
  Mine:
    1. Manager or superintendent
    2. Foremen and shift bosses

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[1] The Buying and Selling of Ores and Metallurgical Products, C. H. Fulton, *T.P.* 83, *U.S. Bur. Mines*; Rukard Hurd, Iron Ore Manual of the Lake Superior District.

3. Engineering staff, surveyor, sampler, assayer, electrician, mechanical engineer, etc.
4. Timekeeper, accountant

Treatment plant:

1. Metallurgist
2. Foremen and shift bosses
3. Chemist and assayer

IV. Miscellaneous

1. Accident or compensation insurance
2. Office expenses
3. Interest on stock of supplies carried
4. Legal expense
5. Consulting engineer
6. Pensions and extraordinary expenses
7. Welfare expense
8. Tax on output
9. Royalties

The difference between income and outgo is used in computing the value of a mineral deposit. The total tonnage multiplied by the net value will give the total profit, and from the total profit the present value can be computed. It is evident that costs must be closely estimated in order to forecast the profit accurately. In a new property an exhaustive study of conditions influencing costs must be made and certain margins allowed to cover unknown or unusual conditions. Experience and actual acquaintance with mine operation are necessary qualifications for estimating costs. Where the mine is in operation, working costs can be determined by a careful analysis of the accounts. Where other mines in the vicinity are in operation much detailed information can sometimes be obtained and a more accurate forecast made.

The final summary of the engineer's ore estimate will be given in terms of gross value and net value for each classification of ore, whether proved, probable, or prospective.

Present Value.—The total profit obtainable from the mining of the ore bodies in a mine is not the value of the mine. If it were practicable to remove all the ore and market it in a very short space of time, the total profit would be a close approximation of the value of the mine; but as this is obviously out of the question the time element must be considered as an important factor in determining value. The rate of mining and the rate of return expected form the two other factors.

The term "present value" can be more clearly comprehended if we imagine it to represent a sum of money which it is proposed to invest in a mine. The sum thus invested is expected to return each year a fixed proportion of the total. The profits of the mine must also be sufficient to repay the capital either in equal annual installments or in the form of an annual sum of money placed at compound interest during the life of the mine. If the first method is followed the equation of present values becomes

$$\text{Present value} + nr \text{ present value} = \text{Total profit}$$

The term n is the number of years required to exhaust the property at a given rate of working, while r is the rate of return expected. The equation is given in simplified form as

$$\text{Present value} = \frac{\text{Total profit}}{1 + nr} \tag{1}$$

If the return of the invested capital is provided for by the method of placing a fixed sum each year at compound interest, the equation becomes

$$\text{Present value} = \frac{\text{Total profit}}{n(\text{annuity rate} + r)} \tag{2}$$

The annuity rate for various rates of compound interest and fixed time periods can be found in engineers' handbooks.

If the life of the property can be accurately determined, the rate of return becomes the most important factor in determining present value. The rate of interest on mortgages ranges from 5 to 6 per cent, and the security given is from 50 to 100 per cent greater than the sum loaned. Mining investments are assumed to involve a greater risk, and as a consequence a higher rate of return is expected. Ore estimates are more or less uncertain in amount, prices vary, markets are sometimes greatly depressed, and the expected profits may be reduced. Equally must be considered the probability of higher prices, greater extraction with improved methods, lower working costs due to increased efficiency, and the possibility of discovering new ore shoots. The element of personal judgment, as well as a careful review of the salient economic features, influences the selection of the rate of return. A minimum rate of return would be from 7 to 8 per cent.

In Table 132 are assumed a total profit of $1,000,000, an interest rate of 10 per cent, and three time periods—5, 10, and 20 years. The present value has been computed for each case and for each equation.

The importance of the time factor is brought out by the comparative figures. This factor is determined by the rate of working. A large annual tonnage requires a larger initial investment for the plant, and while the presumption is that the larger tonnage could be treated at a lower operating cost and thus compensate for the increased cost of the plant, it by no means follows that such expectations would be realized in every case. Too large a plant is as objectionable as too small a plant. It is difficult to lay down any fixed ratio between ore reserves and annual tonnage. Some engineers assume a life of 10 years and plan the annual tonnage accordingly. The difficulty of securing sufficient capital usually restricts the scale of operations.

The present value decreases with a high rate of return and increases with a low rate. There is a considerable difference between the present value as determined by equations (1) and (2) for the same rate of return. The choice between the two results is largely a matter of opinion and to some extent depends on the financial policy of the company. The dividend paid may include the annual allotment for the redemption of the capital invested, and under these conditions equation (1) could be used. If the company sets aside from the annual profit a yearly fund to be invested in income-paying securities for the redemption of the original capital and pays to shareholders as dividends only a given percentage rate on the investment, equation (2) could be used.

TABLE 132

	I	II	III
Total profit	$1,000,000	$1,000,000	$1,000,000
Rate of return, per cent	10	10	10
Life in years	5	10	20
Present value, Eq. (1)	666,666	500,000	333,333
Annual payment	200,000	100,000	50,000
Interest return	66,666	50,000	33,333
Annual return of capital	133,333	50,000	16,667
Present value, Eq. (2)	704,225	545,553	374,251
Annual payment	200,000	100,000	50,000
Interest return	70,422	54,555	37,425
Annuity	129,577	45,444	12,575
Annuity rate, per cent	18.4	8.33	3.36
Return on present value[a] Eq. (1), per cent	30.0	20.00	15.00
Return on present value,[a] Eq. (2), per cent	28.4	18.33	13.36

[a] Annual percentage of present value.

Partially Developed Mines.—The preceding discussion of present value assumes that the ore deposit is in a condition approximating complete development. The principles established could be applied to the developed portion of any mineral deposit. The value of a partially developed deposit is problematical, since both the quantity and value of the undeveloped portion are unknown. The important question to the engineer is the probable extent of the ore bodies. The experience of the engineer and his knowledge of ore deposits, together with a detailed study of the geological conditions of the ore body in question, serve to give commercial validity to his predictions. The engineer must determine whether the development is partial to a limited degree or closely approximates complete development, for in the former the undeveloped extensions may be of far greater importance than the developed ore, and in the latter the probable extensions may be unimportant in their relation to the value of the deposit. While sales of partially developed properties are often made on a lump-sum basis, a conditional sale involving payment of a royalty on ore developed and worked in excess of that shown by the development workings at the time of the sale is more equitable to both parties.

Prospects do not admit of valuation in accordance with any fixed principle. They are sold either on a lump-sum basis or under a working bond. In the latter case a fixed sum is stipulated and the buyer agrees to do a certain amount of development within a given time. The results of the development determine whether the sale is made.

Too many engineers are prone to adopt rigid rules in arriving at the valuation of partially developed mines and prospects, and as a consequence reject properties that eventually become in time active mines. The fact that every prospect is a potential mine until proved otherwise should not be overlooked. Protecting a client from possible loss is highly desirable, but to cause him to lose an excellent property by overconservatism is equally reprehensible. While the risk in purchasing prospects is often great, nevertheless there is frequently the possibility of large profits. Thoroughly developed mines are necessarily sold on narrow margins. Prospectors are as a rule highly optimistic and the asking price of a claim may be prohibitive. The tactful handling of the question of price is essential, and reasonable terms can be arranged in some cases by a just consideration of the interests of both buyer and seller.

Deposits producing minerals that are sold under intensely competitive conditions may well be placed in a separate class. Their value depends on the development of a new market or entry into an established market already fully supplied. Under the latter condition an energetic sales organization is a necessary feature. Time and considerable outlay are necessary before the product can be placed on an equal footing with its competitors. These factors must be taken into consideration in estimating the value of a deposit of this class.

Deposits that show a reasonably large quantity of mineral but values too low to admit of a profit under the existing conditions, or for which a market is absent or where railroad rates to the nearest market are prohibitively high, cannot be termed valueless. The term "speculative value" is applied to such a deposit. Time, developments in ore treatment, expansion of markets, new markets due to increase in population, and increased transportation facilities are the factors that change speculative value into real value. There are no fixed rules for determining speculative value. A careful study of all the conditions that in time might create a market for the mineral, an estimate of the time required, and a prediction of the future selling price might enable an engineer to make a rough approximation of the present speculative value. Prices are nominal in amount where properties of this nature are bought and sold.

Factors of Safety.—An accurate estimate of quantity and average value can be secured where development workings are sufficiently numerous, but in the absence of sufficient development the estimate is approximate.

Where many samples have been taken the average value may very closely approach the actual average value and no discounting factor may be necessary, but under the reverse conditions, some discounting factor is necessary with a small number of samples. Herbert C. Hoover states as the result of his experience that there is a discrepancy between the estimated value and the recovery plus the value in the tailing. In specific examples this discrepancy ranged from 10 to 12 per cent in the Rand gold mines and 12 per cent at three Broken Hill lead mines.[1] Individual factors can be applied to volume, density, average value, selling price, and recovery estimates, each factor being selected after a careful consideration of the conditions that might unduly increase each estimate, or a single factor could be applied to the final estimate of value. Operating and plant costs are apt to be underestimated.

Report.—The result of a mine examination, whatever its purpose may be, is drawn up in a written report. This may be an elaborate or a terse presentation of facts and conclusions. The following outline summarizes the main features of a report:

Name of property—ownership, description
Geographic position—access, climatic conditions, natural resources
Limiting conditions
Geological conditions—type of deposit, structure, mineralization, wall rocks
Development workings
Working plant
History—past production, costs, profits
Ore estimate—average value, proved ore, probable ore, prospective ore
Ore treatment—recoverable values, product, marketing product
Costs—mining and ore-treatment plant
Net profits
Valuation
Conclusions
Locality map; claim map; mine map showing development, ore shoots, and position of samples

Some engineers preface their report with a condensed statement of the important conclusions, thus giving prominence by position to the vital conclusions that logically follow the presentation of the observed facts.

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[1] "Principles of Mining." Herbert C. Hoover, p. 12.

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